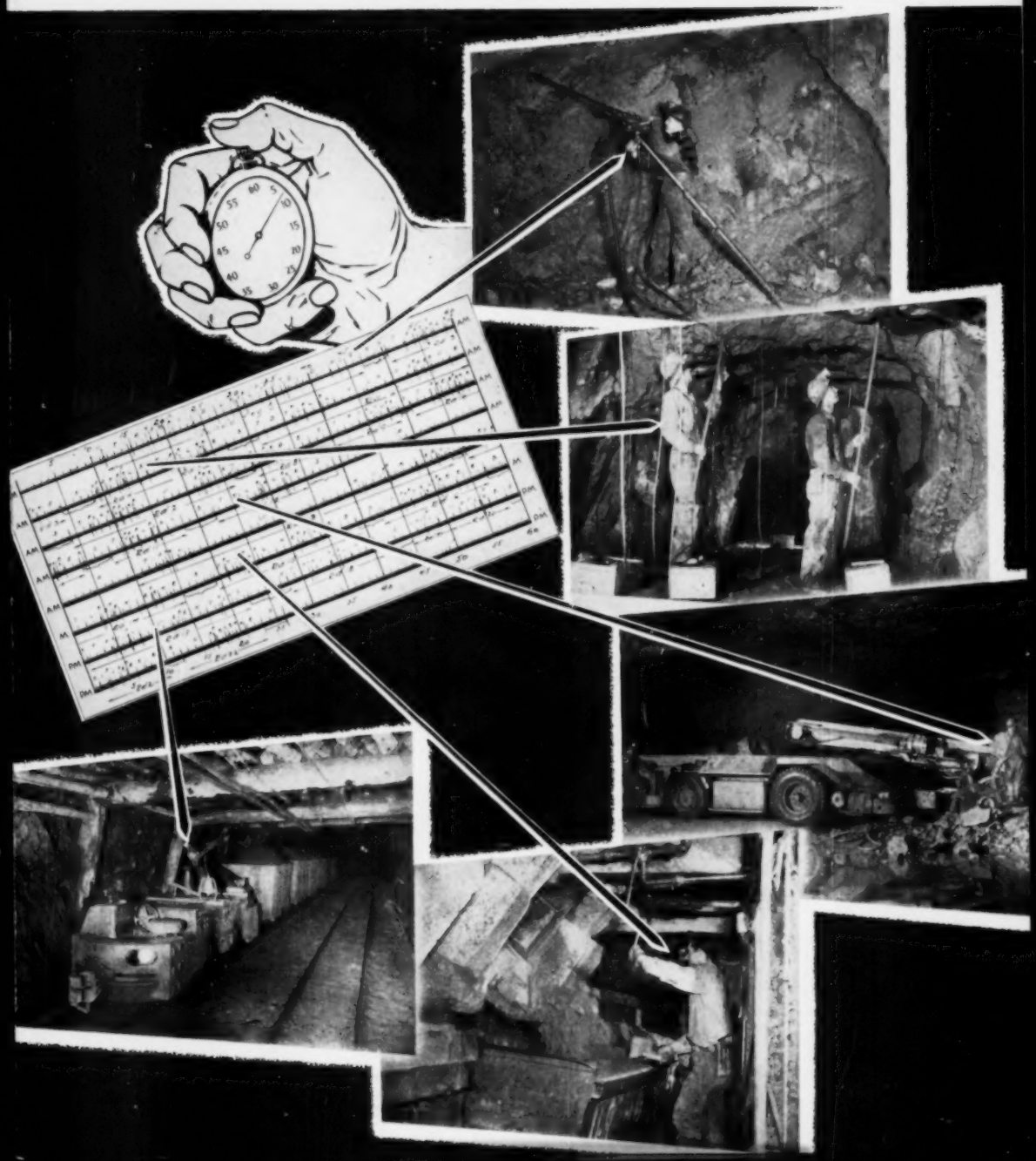
MINING ENGINEERING

SEPTEMBER 1950



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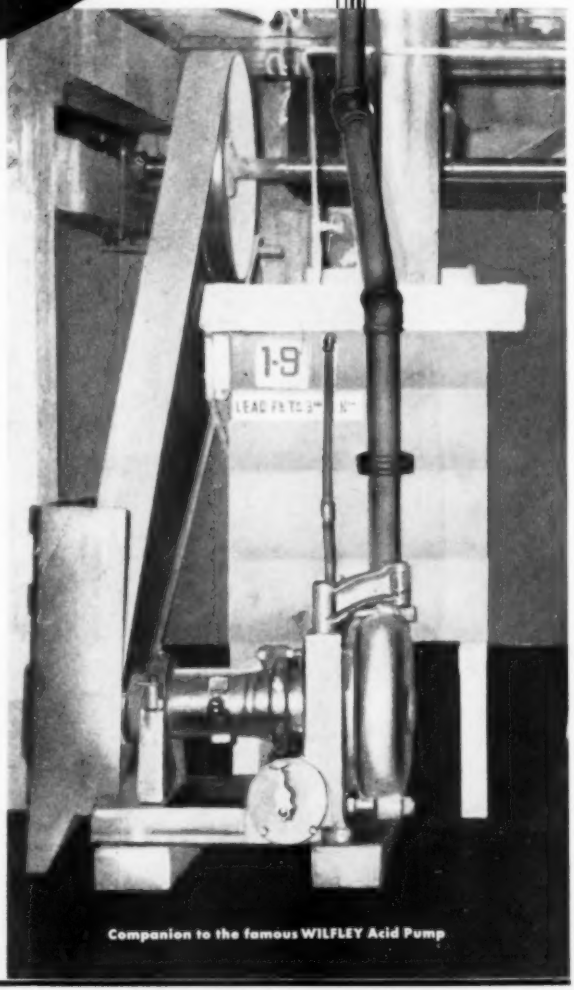

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MINING ENGINEERING

Incorporating Mining and Metallurgy, Mining Technology and Coal Technology

VOL. 189 NO. 9 September 1950

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On the Cover: Time study can be used to increase efficiency of drilling, blasting, cleanup, loading, and haulage. For story of how this was done at Titania see article p. 935.

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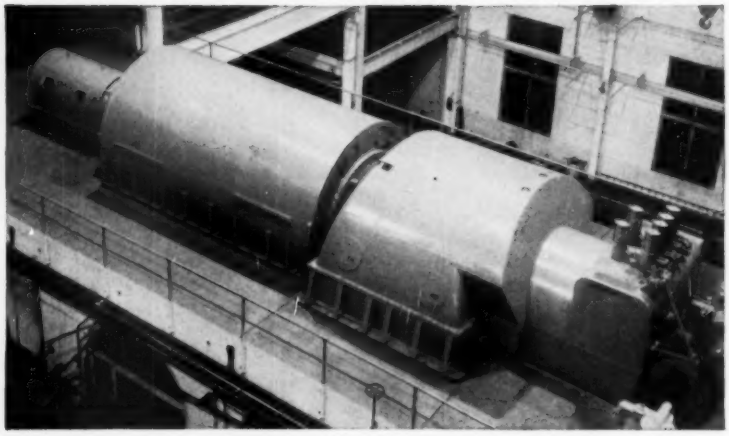
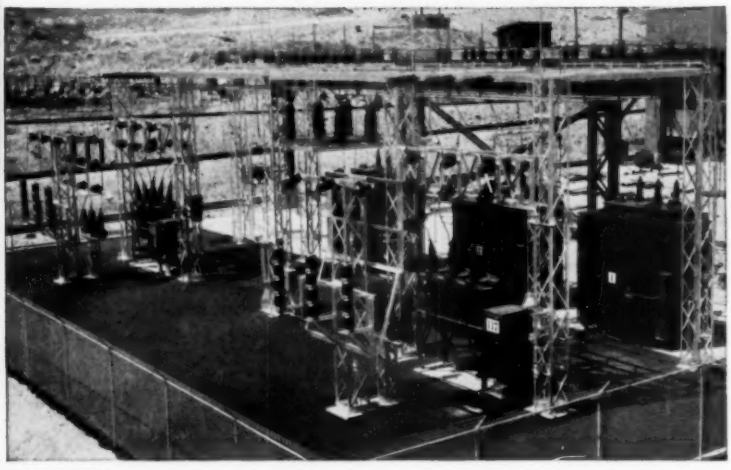
Powered for low-cost high-tonnage refining!

1 Typical of the mining industry's increasing electrification to handle high tonnages at low cost is this new copper refinery at Garfield, Utah, which has a capacity of 12,000 tons of refined copper per month, with provision for expansion to 16,000 tons. Power for the refinery and other operations comes from Kennecott's power station, where three G-E steam turbine-generators (two of 25,000 kw each, plus the 50,000-kw unit shown here) generate 13.8-kv power which is stepped up to 44 kv for transmission.



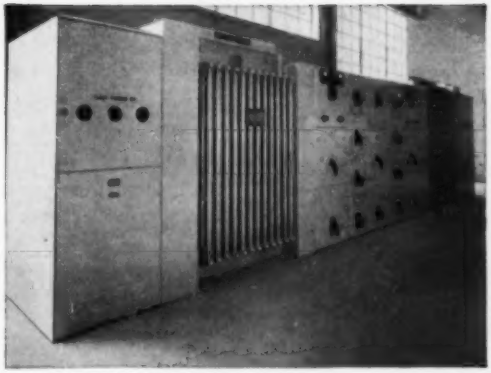


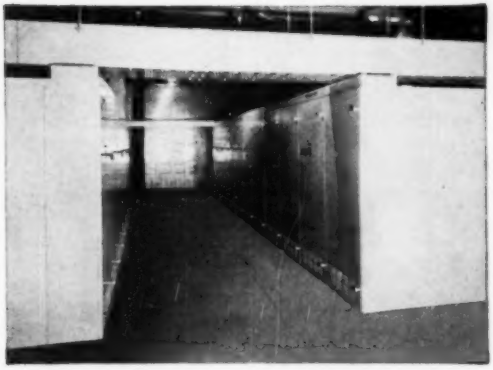
2 At the refinery, three miles away from the power plant, incoming 44-kv power is stepped down to 13.8 kv at this G-E package substation. Completely co-ordinated, it includes an outdoor steel switching structure, three 7500/9375-kva power transformers (with provision for addition of a fourth) and necessary metal-clad switchgear. G-E package substations, made in many standard combinations to fit all mining-industry needs, come in factory-built sections ready to install. They simplify ordering, save engineering time, speed installation, cut costs.

GENERAL ELECTRIC

660-20

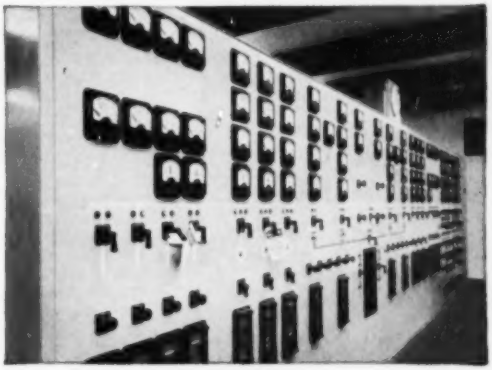
In its new electrolytic refinery, Kennecott Copper Corporation uses this General Electric generation, conversion, and distribution equipment to help maintain production continuity





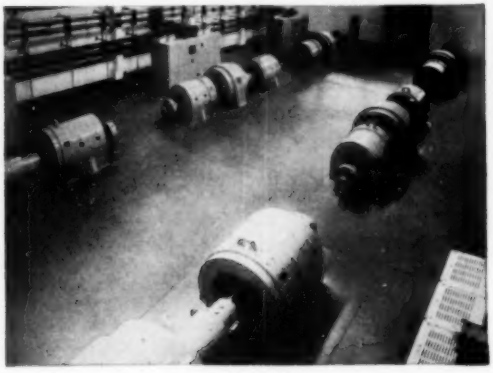
5 Four G-E 1500-kva double ended load-center substations (one shown) step down voltage from 13,800 to 480 and provide distribution at centers of load to reduce power losses.

6 This G-E panel — controlling package substation, a-c switchgear, and d-c power for electrolytic cell lines — helps centralize and co-ordinate plant operations.



3 Inside the refinery, protecting all 13.8-kv feeders and personnel as well, is this G-E metal-clad switchgear that provides flexibility for future load changes.

4 A-c is converted to d-c for electrolytic cell lines by six G-E motor-generator sets, each including a 2900-hp synchronous motor and two 1000-kw, 125-volt d-c generators.



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Mining Engineer, technical graduate. Extensive experience exploration, development, operation and construction. Competent technical reports. Employed. M-574.

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Metallurgist, 34, married, speak Spanish, B.S. degree. Nine years' varied experience sulphuric acid plant operations; mine, mill and smelter chemical laboratories; ore dressing on flotation, gravity and magnetic separations; and milling operations for tin, tungsten and gold. Desires permanent position in U.S. Available immediately. M-578-471-E-12-San Francisco.

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Mining Engineer experienced in using latest technical means to develop and work out manganese mines. Should be familiar with the machineries and from where those can be procured. Appointment will be made for three years. Traveling expenses paid. Location, India. Y3949.

Sales Representative with metallurgical engineering training, 30-35, with thorough knowledge of high grade steels, such as tool steels, etc., and their applications. Must be able to assume complete charge of branch office to be opened in Toronto district, and must have good connections for a Swedish steel manufacturer. Headquarters, New York, N. Y. Y3937.

Assistant Chief Engineer, graduate, with several years' experience in the coal industry, and who is particularly interested in engineering, not operating, for underground mine engineering and surface construction work. Prefer married man. Location, Pennsylvania. Y3936.

Field Geologist, graduate, with three to five years' experience in mining geology in the field. Salary open. Location, Venezuela. Y3928.

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The Dorr Company—Engineers **Stamford, Connecticut**

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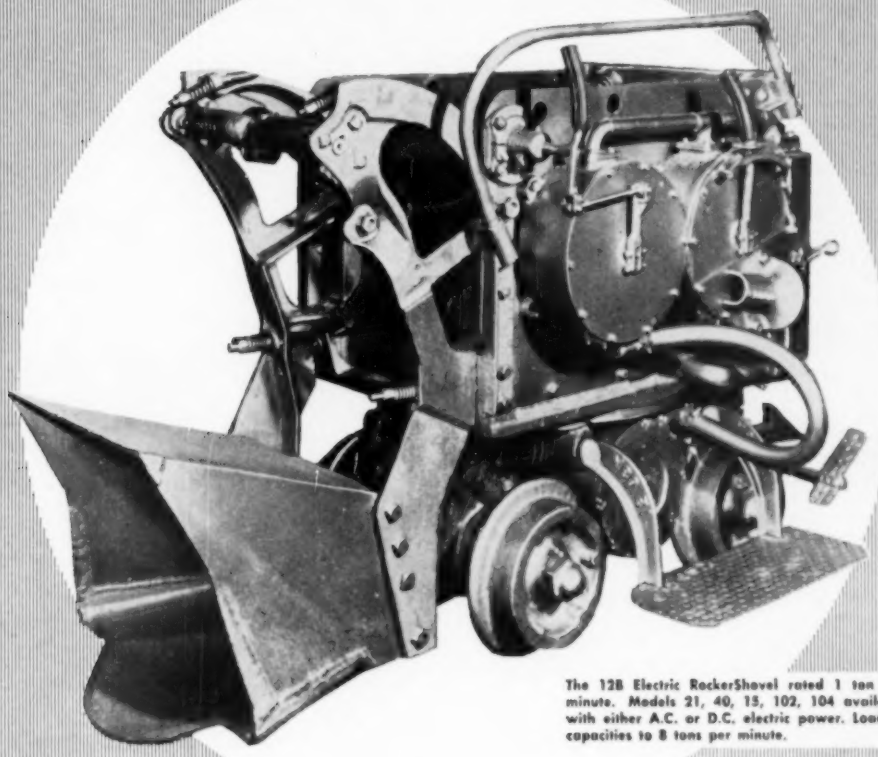
Metallurgical engineer, with several years of milling experience, preferably in flotation and cyanidation, to make laboratory tests and occasionally to operate pilot plant on a wide variety of ores under only general direction of Chief Metallurgist. Successful candidate must be willing to make this type of work his career and must be able to get along well with others. Location—the laboratories of The Dorr Company, Westport, Connecticut. Salary to start, $400.00 per month. Apply with photo and complete resume of education and experience to: Personnel Director, The Dorr Company—Engineers, Stamford, Connecticut.

EIMCO 12B Electric RockerShovel



The 12B Electric RockerShovel rated 1 ton per minute. Models 21, 40, 13, 102, 104 available with either A.C. or D.C. electric power. Loading capacities to 8 tons per minute.

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The Eimco 12B machine is the smallest loading machine on the market with clean-up range, speed, power, stability and operating ease far superior to other machines in its class. The 12B Electric is the only small machine offered with electric power and duplicates its air powered relative in all the desirable characteristics of a loading machine.

The Electric 12B is available for A.C. power and with either permissible or non-permissible design. Overall caging dimensions are practically the same as the air driven machines: length (bucket up) 42", width (including controls) 37".

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Authors in This Issue

C. D. Hoyt (P. 935) has just returned from Europe where he spent a year studying at the Norges Tekniske Hogskole and then traveling through Norway, Germany, Italy, Austria, Switzerland, France, Spain and England, studying mining methods under a Fulbright scholarship. A graduate of Penn State, he has also studied at Columbia and R.P.I., and has worked at the Island Creek Coal Co. and the Westmoreland Coal Co., in West Va. and Pennsylvania, respectively. A New Jerseyite by birth, Mr. Hoyt now lives in Montoursville, Pa.

George T. Bator (P. 941), associate professor of mining engineering at the University of Alabama, took his M.S. from that school in 1941. After earning his B.S. at Washington State in 1938, he went to work for Howe Sound, and after two years took time out to work for his advanced degree. Then he returned to Howe Sound, but the next year was back at Alabama as an instructor in mining.

S. T. Harrison (P. 943), chief engineer for the J. L. Shiely Co. in St. Paul, has been in engineering for nearly 40 years, working on construction problems, safety, operating limestone quarries, and helping build the Ft. Peck, Sardis, and Garrison Dams. He served in the Navy in both World Wars, as a lieutenant and then a captain in the U.S.N.R. Mr. Harrison was born in Duluth, attended Michigan College of Mines and Technology, Northwestern, and LaSalle.

V. F. Parry (P. 974) is now conducting research on western coals, including gasification, carbonization, drying and briquetting and combustion, for the Bureau of Mines in Denver, as chief of the coal branch for Region IV. After taking his M.S. from Carnegie Tech in 1923, he started as head of coal research for Combustion Utilities Corp. in New York, remaining there for twelve years. Then he became supervising engineer of the subbituminous and lignite station of the USBM in Golden, Colo., and since 1949 has held his present post. Mr. Parry hails from Ogden, Utah, attended Weber Academy and the University of Utah. He is an AIME member.

C. G. Fink (P. 952) has headed Columbia University's department of electrochemistry since 1922, and has also carried on his own consulting practice at the same time. He holds a B.A. degree from Columbia ('03), took his Ph.D. at Leipzig in 1907, and was granted an honorary Sc.D. from Oberlin in 1936. Dr. Fink has worked for the General Electric Co., the Edison Lamp Works, and headed Chile Exploration's research laboratories in New York. He was the originator of the drawn tungsten filament, platinum substitutes for tungsten lamps and radio tubes, and other noteworthy developments.

E. O. Wagner (P. 974) is a chemical engineer with the coal branch of the Bureau of Mines at Denver, and has been there since 1942. He began his career with the Bureau of Standards in that city. Born in Neustadt a.d. Haardt, Germany, he attended high school in Longmont, Colo., and graduated from Western State College, Colo.

G. L. Putnam (P. 952), a chemical engineer, has been a research associate at the University of Washington for the past three years. He has spent a year as an associate professor at Oregon State, and spent four years with the Carborundum Co. in Niagara Falls. Dr. Putnam took his M.S. from the University of Washington in 1938, and his Doctorate at Columbia four years later. He holds numerous patents, and is a prolific technical writer.

V. F. Parry

E. O. Wagner

A. J. Alexander (P. 967) has worked in the coal mines of West Virginia for thirty years, in every capacity from that of a day laborer to owning and operating coal mines. For many years he was superintendent for the Island Creek Coal Co., and is now chief of the West Va. department of mines in Charleston. He was born in Charleston and attended Washington and Lee University and the University of Illinois.

L. C. McCabe (P. 971) is chief of the office of air and stream pollution for the Bureau of Mines. He headed the Bureau's coal division in 1946-7, and from 1947-9 headed the Los Angeles County air pollution control district. Born in Graphic, Ark., he went to high school in Fort Smith, and then on to the University of Arkansas, Northwestern, and the University of Illinois for B.S., M.S., and Ph.D. degrees. Dr. McCabe began his career as a geologist for the Mississippi Coal Corp., and spent 11 years with the Illinois Geological Survey. Entering the Army in 1941 as a Captain, he rose to the rank of Colonel in the Corps of Engineers and the General Staff Corps, served as chief of utilities, and chief of the German and Belgian solid fuels sections, SHAEF.

T. C. King (P. 951) is a metallurgist at the Central Mill of the Eagle Picher Mining and Smelting Co. in Cardin, Okla., and has been with that company for four years. Mr. King started his career as a mining engineer with the Cia. Minera de Guazapares, Mexico, in 1935, and 4 years later returned to the States to take his degree in metallurgical engineering at the University of Arizona. The Navy then claimed him for 3 years, and he served as a lieutenant j.g. on a CVE in the Pacific. Mr. King lives in Baxter Springs, Kansas, and is a Junior Member of Aime.

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* Pellets of concentrated iron ore have been smelted in the blast furnace by Armco and the results proved satisfactory. Various percentages of pellet burdens were tried and it has been found that charges up to 70 pct pellets are workable.

* In the reorganization of the Department of the Interior, two new assistant secretaries were appointed and the work divided between the four assistant secretaries. C. Girard Davidson will head a division of mineral resources including the Bureau of Mines, Geological Survey, Division of Oil and Gas and Division of Geography.

* Germanium, one of the rarest of metals and a wartime strategic material, has been discovered almost in the shadow of the Capitol. The only catch is that the scattered nature of the deposits makes profitable recovery extremely doubtful. Nevertheless, experts have tabbed the concentration in the District of Columbia area the world's second richest known concentration of the element.

* Cakes of mica, weighing up to 500 lb, have been formed at the Colorado School of Mines by a new "cool hearth" method. The process avoids use of crucibles and contains the molten mica in a basin of unmelted raw mixture from which the melt was derived. Crystallization takes place in this melting basin which forms a center pool surrounded by a solid cool mass of the unmelted raw mixture.

* William J. Baxter, of the International Economic Research Bureau, predicts a period of deflation because of war-essential restrictions on consumer credit and materials use. He stated; "the recent prosperity was based mainly on the sales boom in autos, homes, and household accessories."

* Two subsidiaries of United States Steel have decided to build four new ships immediately to augment iron ore and limestone carrying capacity on the Great Lakes to meet demands created by the national defense emergency.

* The Norfolk and Western Railway is making a complete change over to coal to drive its locomotives.

* A national roster of scientific personnel similar to the one used in the last war will be established by the Office of Education, by agreement with the National Security Resources Board. The new unit, the National Scientific Register Project, will make studies of the Nation's scientific manpower supply and recommend steps to increase the numbers of skilled personnel where needed.

* National Lead Co. will start construction immediately of new facilities at St. Louis, Mo., for production of titanium pigments for paints. Producers of titanium pigments have been under considerable pressure to meet all demands from paint manufacturers, particularly for titanium dioxide.

* The basic criteria for deferment of reservists in civilian occupations are (1) the Department of Labor List of Critical Occupations, and (2) the Department of Commerce List of Essential Activities, both of which were announced Aug. 3. Fundamentally, the reservist deferred must be engaged in a critical occupation which is necessary to a highly essential activity, and he will be deferred only until he can be satisfactorily replaced in that occupation. A delay in call to active duty, it was emphasized, may not be considered as an indefinite exemption from military service.

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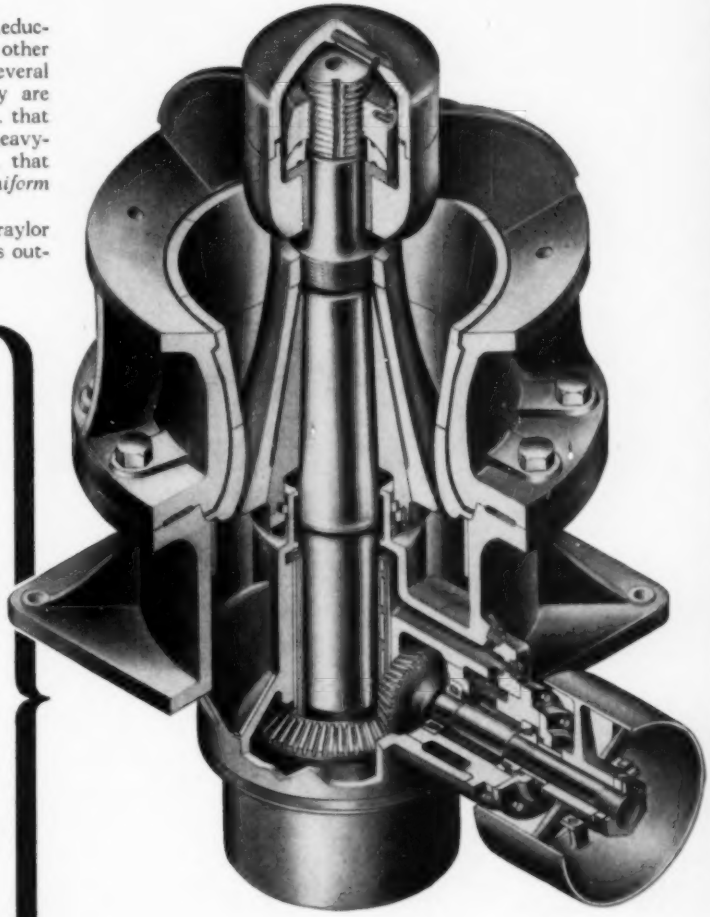

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Traylor's curved concaves and bell head apply power as a *direct* crushing force. The eccentric motion of the bell head exerts power *directly counter* to the curved surfaces of the concaves. This stops lifting and churning of material . . . cuts down power loss . . . reduces power costs per ton.

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Since there is no choking or packing . . . *no lifting and churning* . . . material flows evenly thru the crushing chamber of a Traylor TY Reduction Crusher. This reduces the percentage of waste fines. The curved crushing surfaces produce a more uniform product at less cost per ton.

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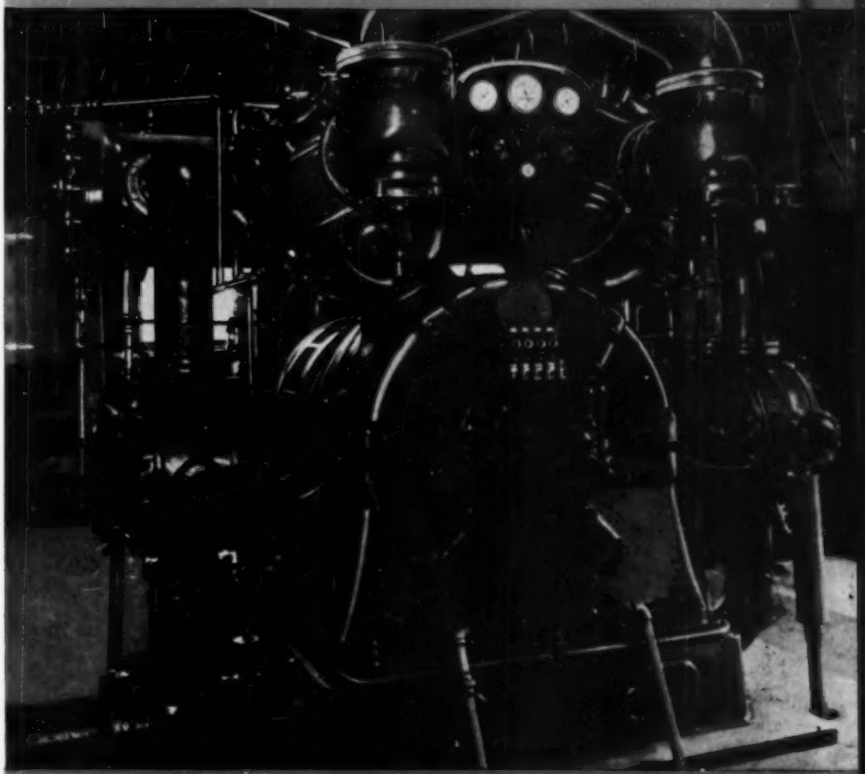
- ★ **Simpler Design—Longer Life—Less Maintenance**

JOY WN-112, two cylinder, two stage, double acting, is a heavy duty, continuous type compressor, built in capacities from 368 to 1828 CFM, and up to 3656 CFM as twin units.



JOY WN-114 in various sizes delivers from 1092 to 3656 CFM, and up to 7312 CFM in twin units. It's a heavy duty, continuous type four cylinder unit; two stage, double acting.

Write for Bulletins, or

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W&D M 2828

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ORE BENEFICIATION

Through A-C Equipment and Engineering

FOREIGN ORE DISCOVERIES ARE GOOD NEWS, but domestic mines still will supply the bulk of the ore consumed in the U. S. Problems of transportation and availability will assure this. *One thing certain* in this changing picture, more mining, processing and handling will be required on domestic ores as quality lowers.

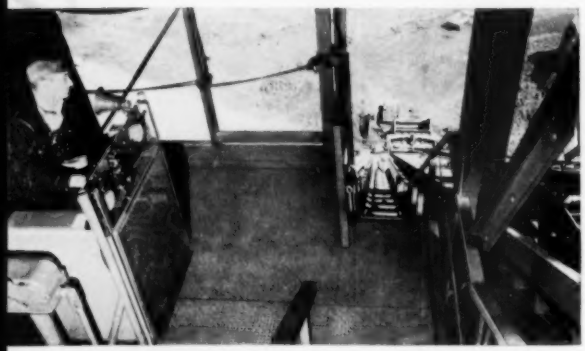
In iron mines throughout U. S., Allis-Chalmers equipment for half a century has been processing ores for the steel industry. Its research, engineering and experience is today helping ore producers meet the challenge of economical beneficiation of leaner ores and taconites.

Operation photos, below and right, show A-C ore beneficiation equipment now in wide use. Allis-Chalmers offers you this kind of *undivided responsibility* for process, drive and control equipment all along your flow sheet.

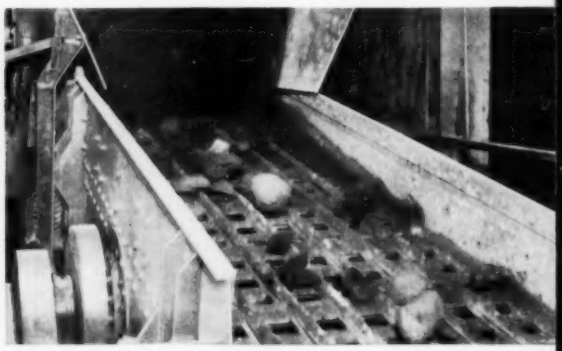
A-C also operates a complete commercial research laboratory and pilot plant to aid you in developing new processes. For detailed information on equipment and services, call your nearest A-C office. A-3095

ALLIS-CHALMERS, 971A SO. 70 ST.
MILWAUKEE, WIS.

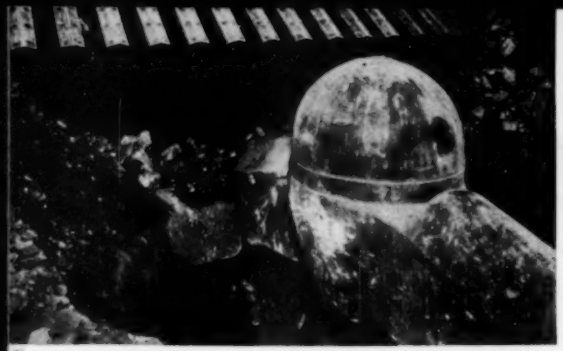
Texrope, Hydrocone and *Low-Head* are Allis-Chalmers trademarks.



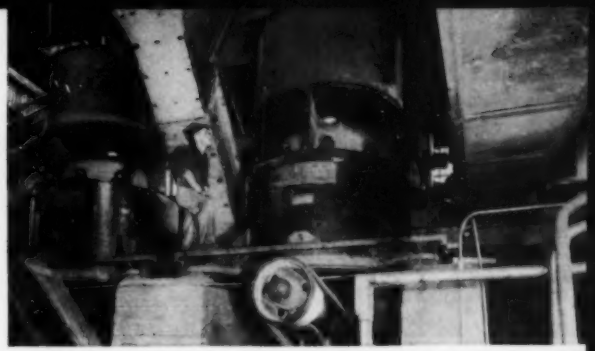
1 **SINCE UP TO THREE TONS** of ore are needed to produce one ton of concentrate, large quantities of low grade ore must be removed from the deepening mines. Mesabi range mine skip hoist above is A-C, built by the *only* manufacturer of hoists and *all* auxiliaries.



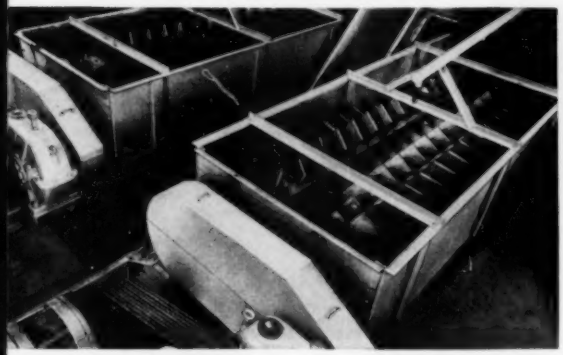
2 **FINES ARE OFTEN SCALPED** from run-of-mine ore on sturdy 1000 tph Style C vibrating screens like this one at a Minnesota iron mine. Allis-Chalmers builds *six* separate types of vibrating screens to meet varying beneficiation requirements.



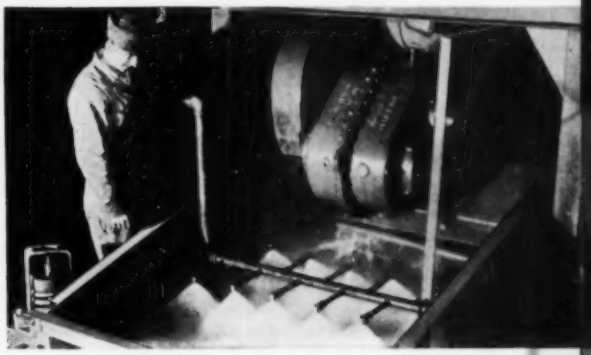
3 **TOUGH, ABRASIVE, LOW GRADE** iron ores are crushed in high output primary gyratory and jaw crushers. This 60-in. Superior-McCully crusher is in use in the Lake Superior region. Write for Bulletin 07B6004 for complete details on this type of crusher.



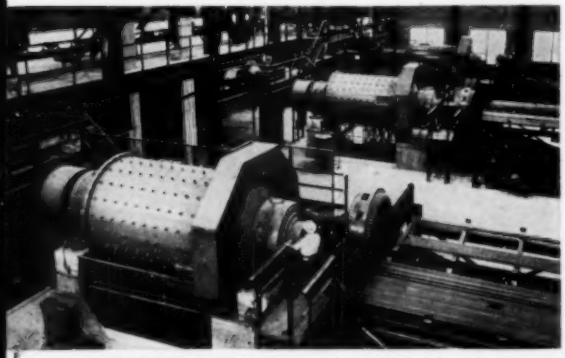
4 **FINE REDUCTION** of the iron ore is often accomplished with *Hydrocone* gyratory crushers. Two are seen working in parallel at this Adirondacks iron mine and beneficiation plant. Write for Bulletin 07B7145 for complete description and operating data.



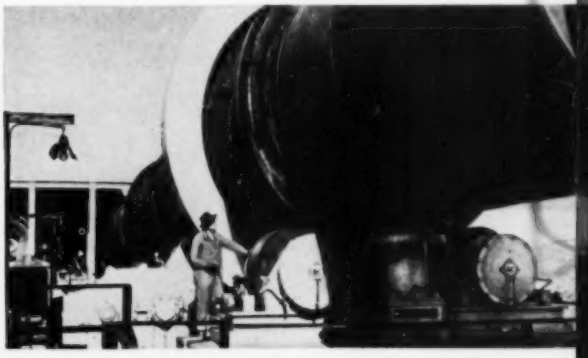
5 **THESE LOG WASHERS** are used to remove silica and medium sticky clay from crushed iron ore. Contra-rotating steel paddles produce a scuffing, scouring and sluicing action. Allis-Chalmers makes all basic types of washing equipment. See Bulletin 07B6411.



6 **IN CONCENTRATING IRON ORE** by the heavy density process, *Low-Head* vibrating screens are widely used for recovering media. The first screen section drains the media while the second section washes at this Minnesota plant. Write for Bulletin 07B6330.



7 **IRON ORE IS FINELY GROUND** in three A-C 8 ft x 12 ft rod mills ahead of magnetic separation at this Adirondack plant. A-C builds *five* separate types of grinding mills to meet varying plant requirements. Request Bulletin 07B6718 for full details.



8 **NODULIZING OF BENEFICIATED IRON ORE** so that it can be used efficiently as blast furnace feed is the job of this Allis-Chalmers 11 ft, 6 in. x 300 ft rotary kiln at a southern iron mine and beneficiation plant. Bulletin 07B6368 contains full description.

EQUIPMENT FOR IRON ORE BENEFICIATION: Hoists . . . Jaw and gyratory crushers . . . Vibrating screens . . . All types of washers . . . Concentrating jigs . . . Grinding mills . . . Rotary kilns and coolers . . . Centrifugal pumps . . . *Texrope* V-Belt drives and sheaves . . . All types and sizes of Electric motors . . . Starters . . . Complete power generation, distribution, control equipment. Write for literature.

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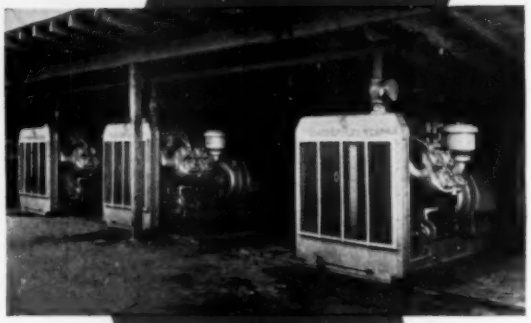

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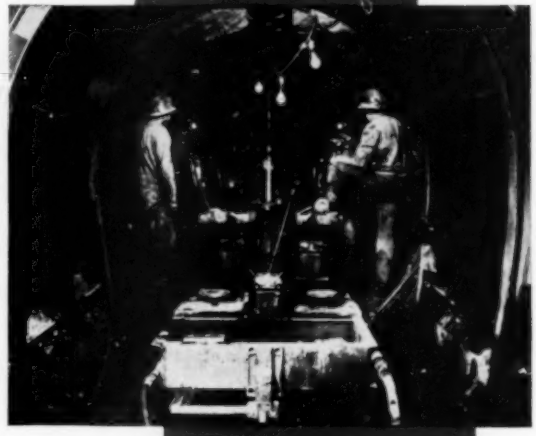

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The Taconites Are Ready

POLITICAL developments, the impact of Korean warfare necessitating immediate rearmament plans, and recent developments in mining taconites have focused attention on their immediate exploitation to fill the gap created by diminishing reserves. The question of rich foreign ores versus low grade development is past—we need both sources with emphasis on the immediate large scale mining of taconites.

In the past three months several things have occurred which add to the taconite picture:

(1) Pellets have been used successfully in the blast furnace by Armco.

(2) In Congress HR 8512 was introduced by Congressman Blatnik of Minnesota to amend the lending powers of RFC to permit long term loans on basic mining developments, particularly the taconites, which may be needed for the strengthening of the national economy.

(3) Walter S. Tower, president of the American Iron & Steel Institute, announced that American steel companies are planning expansion of 6,000,000 tons of steelmaking capacities in the next few years.

(4) The outbreak of war in Korea has obliged the United States to rearm and make plans for a semipermanent defense establishment to which 10 to 12 pct of the national income will be budgeted.

(5) Startling new advances have been made in the Linde Air Products Co.'s jet-piercing drilling.

The United States is now confronted with the problem of increasing its steel capacity and at the same time of securing enough iron ore to supply this increase as well as make provision for the rapidly decreasing supply of rich iron ore on the Mesabi range. Taconites, the low grade iron ore at the east end of the Mesabi, of which there are literally billions and billions of tons of easily concentratable magnetic ore, are the immediate answer.

According to the *Wall Street Journal* on July 24: "Reserve Mining Co., backed by such steelmakers as the Armco Steel Corp., Wheeling Steel Co., Republic Steel Corp., is reported ready—as soon as the pending bill clears (HR 8512)—to seek a $100 million federal loan to build a taconite processing plant at Beaver Bay, Minn., to produce about five million tons (of pellets) annually of 60 pct to 65 pct in iron ore . . ." Experimental work indicates that pellets may be the final form, rather than nodules, agglomerates, or sinter. At Ashland, Ky., the Reserve Mining Co., which has obtained a

(Continued on next page)

(*Continued from preceding page*)

lease from Mesabi Iron Co. on its holdings, built a pelletizing plant which has been in operation since last spring. Sufficient pellets have been accumulated to make test runs in the blast furnace, starting with low percentages. Recent charges up to 70 pct of pellets have been handled in the blast furnace satisfactorily.

The Pickands Mather & Co., in which Bethlehem Steel and Youngstown Sheet & Tube have a large interest, formed the Erie Mining Co., to mine the taconites at the west end of the Mesabi Iron Co. holdings at the east end of the Mesabi Range. Here, an experimental mill of 2000 tons daily capacity has been operating for more than a year to study the various production problems of crushing, concentrating, and pelletizing. The results of these studies will be used for the design and location of a large scale taconite processing plant.

Shipments of pellets have been made to the Zenith furnaces of the Interlake Iron Co., but smelting results are not yet available. At Lebanon, Pa., studies on pelletizing are being conducted by Bethlehem Steel Co.

One of the factors which makes the mining of taconites today economic and feasible has been the development of the Linde Air Products Co.'s jet-piercing drill. The magnetic taconites are extremely hard and in the old days no more than 10 to 15 ft a day could be drilled with a churn drill. Jet-piercing can now literally burn holes in the taconites at an average speed of 17 fph, but improvements in burner heads will probably increase this figure. Present 6-in. dia jet pierced holes are a large percentage +8 in. dia. Jet-pierced reaming of these holes, presently under study, will make it possible to use an 8-in. explosive charge instead of the present 6-in. A vastly improved breakage can be expected from these larger holes with greater powder content than the early 6-in. holes which gave around 30 tons per ft of hole. A complete test unit of the Linde Air Products Co. jet-piercing drill has been in operation at the Erie Mining Co.'s taconite operation for the past year or so. This remarkable innovation in drilling hard ground, still being improved for more effective performance (deeper and larger holes), is no less revolutionary in the taconites than was flotation in the case of the porphyry coppers. No cost figures of jet-piercing have been published, but it is reasonable to expect something less than $3.00 per ft. Due to the speed of drilling, however, the Linde device can perform the equivalent work of ten churn drills in a single shift. This means some large indirect economies in operation such as being obliged to provide housing for one-tenth of a drilling force, a large capital outlay in a region like northeast Minnesota, where there is no existing housing for the new personnel. Jet-piercing is revolutionary in its impact on lowering mining costs of taconites, and with negligible stripping, the taconites of Reserve Mining Co. and Erie Mining Co. should be mined at no greater cost per ton than the open pit porphyries; namely, 25¢ per ton.

It is only possible to speculate as far as costs of producing taconite concentrates are concerned; presumably they will not exceed production cost per unit of iron content of product of present underground ores. For an estimate of costs and profits, reference is made to the Congressional Record on May 23 wherein a letter from Arnold Hoffman is entered by the Honorable John A. Blatnik. The following is a quotation from a portion of the letter:

"The estimate in Table I is based on an assumed concentrating ratio of 2.75 tons crude ore to 1 ton of pellets or sinter and costs determined from presently available data and figures of comparative open pit operations.

"This $1.40 profit figure makes no provision for a possible premium on pellets. A substantial premium is possible, however, since pellets are preferable to direct shipping ores in the furnaces. Not only is the blast furnace provided with a selected feed, hard and higher grade, resulting in greater unit production per ton of charge with consequent lower cost, but the introduction of pellets in the open hearth can reduce scrap requirements to a level where "home" scrap from fabricating plants should meet such needs. Pellets make an excellent substitute for scarce lump ore, and with other advantages can easily command a premium from the hard pressed steel industry. In this event, profit estimate submitted might be revised upwards.

"It must be remembered, also, that there is an important saving in the transportation of pellets containing 65 pct Fe as compared to the lower grade direct shipping ores."

The capital investment for development of taconite is admittedly large, $15 to $20 per ton of pellets, but this compares with the cost of building a fleet of ore vessels and other development work in isolated or uninhabited areas to supply the needs of our steel industry from abroad. Especially with the uncertain international picture and the threat of war, it is of prime importance to the security of the United States that an adequate supply of iron ore be developed within our own borders. The taconites are ready to meet this need.

Table I. Estimated Cost and Profit of Blast Furnace Feed from Taconite

Mining & milling per ton concentrate, 2.75 x 65¢		$1.79
Pelletizing per ton concentrate		1.00
Local and state taxes per ton concentrate		.15
Royalties at 8¢ per ton crude ore, 2.75 x 8¢		.22
Haulage, Babbit to Beaver Bay, 2 tons at 1¢ per ton mile, 55 miles... (Up to 35 pct gangue eliminated at mine)		1.10
Amortization at 18¢ per ton crude ore, 2.75 x 18¢		.50
Haulage, Beaver Bay to Ohio port		1.16
Total cost per ton pellets		$5.92
Value ton 51 pct Fe ore at Ohio port	$7.70	
Value ton 65 pct Fe ore at Ohio port at 15.1¢ per unit above 51 pct	9.81	
Operating profit per ton pellets	$3.89	
Operating profit per ton crude ore	1.40	

It's Everyone's Business

National Minerals Advisory Council

A meeting of the National Minerals Advisory Council on August 3rd in Washington, D. C., indicated the vitally important part that the mining industry is to play in the mobilization program. Director James Boyd of the Bureau of Mines told the Council that the Department of the Interior would review the recommendations of all the Council's commodity committees with regard for mobilization planning in the light of the changed international picture. The Council was requested to reactivate its commodity committees and have them gather all available data on supplies, their sources and availability and present and potential production of the minerals and metals represented on each committee. Data on labor, machinery, transportation, automotive and stationary equipment, power, fuel, lumber, water supply are a few of the important items called for in the reports, which are to be presented at a meeting of the Council on September 1 at Salt Lake City. The material in the reports will become the basis for discussing metal and mineral requirements at that time.

Discussion at the meeting bared several $64 questions, probably the most important of which are the following:

1. Which of the war-essential metals and minerals and in what quantities can we reasonably expect to get them from abroad under threat of submarines?
2. How are we going to meet the manpower problem posed by (a) migration of labor from mining to manufacturing since the end of World War II and (b) the draft and the calling up of reservists?

Opinion was expressed by industry spokesman at the meeting that the function of complying with mobilization requirements be left to those in the industry itself; that is, those having the "know how." This view contended that any administrating governmental agency should be kept as small and streamlined as possible.

There was general sentiment against the reactivation of the wartime Premium Price Plan or other bonus plans as a stimulus to production.

The thought was emphasized that what was needed was a change in the basic conditions which have fostered the decline in domestic mining activity in the postwar years. One such condition, long overdue for correction, is the tax structure as it applies to mining enterprises. Many quarters both in industry and in government favor tax relief along the lines suggested in the six tax recommendations by the Council to the Secretary of the Interior last December. The Council adopted a resolution expressing a feeling that the following tax recommendations are still feasible and desirable and will accomplish as much toward increasing exploration for new deposits (thereby subsequently increasing production) as will government loans for exploration: (1) Losses from unprofitable ventures should be allowed corporations, partnerships, or individuals as ordinary deduction against current income. (2) Development costs after discovery should be recognized as operating expenses. (3) Allowance for depletion should be made to the stockholder as well as to the corporation. (4) Income should not be taxed without full allowance for losses of loss years. (5) Adequate allowances for percentage depletion should be made.

A discussion of the manpower problem led to the Council's acceptance of a resolution advising that "military authorities should proceed with caution in depriving the mining and metallurgical industry of its manpower." The resolution strongly urged that no personnel "directly engaged in exploration, development, production or supervision (of strategic and critical materials) should be drafted for the armed forces, at least until the anticipated demands upon these producers are clarified."

Stockpiles

The Munitions Board's "Stockpile Report to the Congress" of July 23, 1950 revealed: (1) The total estimated value of the stockpile objective is $4,051,-714,510 at the close of fiscal year 1950. (2) The total value of the stockpile on hand, at the close of fiscal 1950 was $1,556,154,352 or 38.4 pct of the total stockpile objective. An additional $494,948,060 was on order, making a total of 50.6 pct on hand plus the amount on order. (3) Materials obtained for the stockpile by the ECA from January to June 1950 amounted to $13,112,085, while development projects by ECA during this period involved the expenditure of $9,322,000, mainly with counterpart funds.

Shortly after the start of the Korean conflict it was felt that Congress ould appropriate greatly increased sums for the purchase of materials for the stockpile. This stimulus to the program may increase the dollar earnings of those European nations that are present or potential contractors in our stockpiling program. Such a development would mean that these nations could add to their gold reserves, thereby stabilizing their respective economies and hastening recovery. This seems to be the picture for the next six months anyway. The "bug" appears when it is realized that the increased threat of total world war actually may retard recovery in Europe as nations on the continent may feel inclined to devote more of their resources to defense programs.

Industries Essential to Defense

The Department of Commerce in response to a request by the Department of Defense issued on August 3, 1950 a "Tentative List of Essential Activities" as a "guide for calling up for active duty members of the civilian components of the Armed Forces." The list includes the following: **Primary Metal Industries.** Included herein are establishments engaged in the smelting and refining of ferrous and nonferrous metals from ore, pig, or scrap. **Metal Mining.** This category includes establishments primarily engaged in mining, developing mines or exploring for metallic minerals (ores). This group includes all ore dressing and beneficiating operations. **Anthracite Mining, Bituminous Coal and Lignite Mining, Crude Petroleum and Natural Gas Extraction, Mining and Quarrying of Nonmetallic Minerals, Except Fuels.**

Challenge to the Mining Industry

The source of our country's great strength lies in its capacity to produce. In times of stress such things as national morale and manpower are all-important but without a capable industrial machine we would be helpless. That machine must be fed with minerals and metals in order to generate and maintain momentum sufficient to insure success. Consequences of the lack of adequate supplies of essential metals and minerals to increase and sustain our industrial power are not pleasant to contemplate. It is absolutely imperative that we put forth Herculean effort to guarantee ample supplies of such essential materials as copper, lead, zinc, manganese, antimony, mercury, tungsten, tin, chromite, nickel, cobalt, iron ore and rubber.

The mining industry faces a challenge more serious than ever existed before in the history of our country. The industry must be equal to the task.

CURRENT REAGENT PRACTICE ON COPPER ORES

For the efficient treatment of non-refractory copper ores, Aerofloat* Flotation Reagents have long been favored by many leading producers because of their high selectivity, wide tolerance to changes in feed mineralization and the small amounts required to achieve optimum recovery.

Less well known is the use of Aerofloat Flotation Reagents (alone or in combination with other promoters) to recover copper values from refractory ores, especially where the selectivity of the reagents is of prime importance.

Data on the reagent combinations used for copper recovery have been reported from time to time in technical papers given before ore-dressing groups throughout the mining world. But much of that data is no longer current nor conveniently available to individual mill metallurgists and managers.

*Aerofloat is a registered trademark of American Cyanamid Co. applied to certain flotation reagents of its manufacture.

The Cyanamid Mineral Dressing Laboratory will soon issue an 8-page Technical Bulletin containing interesting resumés of the application of Cyanamid Reagents to a variety of copper ores in Canada, Mexico, South America, Africa, Australia and the United States.

Described are diversified applications of Cyanamid Reagents to ores having such mineral associations as:

Bornite with Chalcopyrite
Chalcopyrite with Pyrite
Chalcopyrite associated with Sphalerite, Pyrrhotite and Pyrite
Chalcopyrite with Magnetite and Pyrite
Chalcopyrite with Galena and Sphalerite
Bornite with Chalcopyrite, Enargite, Tennantite and Chalcocite
Chalcopyrite, Covellite with Galena, Pyrite, Siegenite, Bravoite and Marcasite

Reagent combinations and feed-rates are given in detail with explanatory comment. Examples include both large and small operations and ores with values as low as 0.60% Cu.

Presenting timely, helpful data on the efficient use of Cyanamid Reagents is an integral part of our complete service to mills the world over. It is a logical extension of the work of the Cyanamid Mineral Dressing Laboratory in the creation of new reagents and the development of flow-schemes for the efficient treatment of difficult ores. It supplements the work of Cyanamid Field Engineers in the practical application of our reagents and processes on your mill.

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☐ Kindly send me a copy of your forthcoming booklet, "CURRENT REAGENT PRACTICE ON COPPER ORES"
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because BLACK & DECKER uses Nickel Alloy Steel for extra stamina

Examine this "machine gun of industry" drilling shot holes in a mine . . .

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The Black & Decker Manufacturing Company of Towson, Md., use nickel alloy steels to attain long service life . . . and correspondingly less maintenance, fewer replacements, and lower non-productive labor costs.

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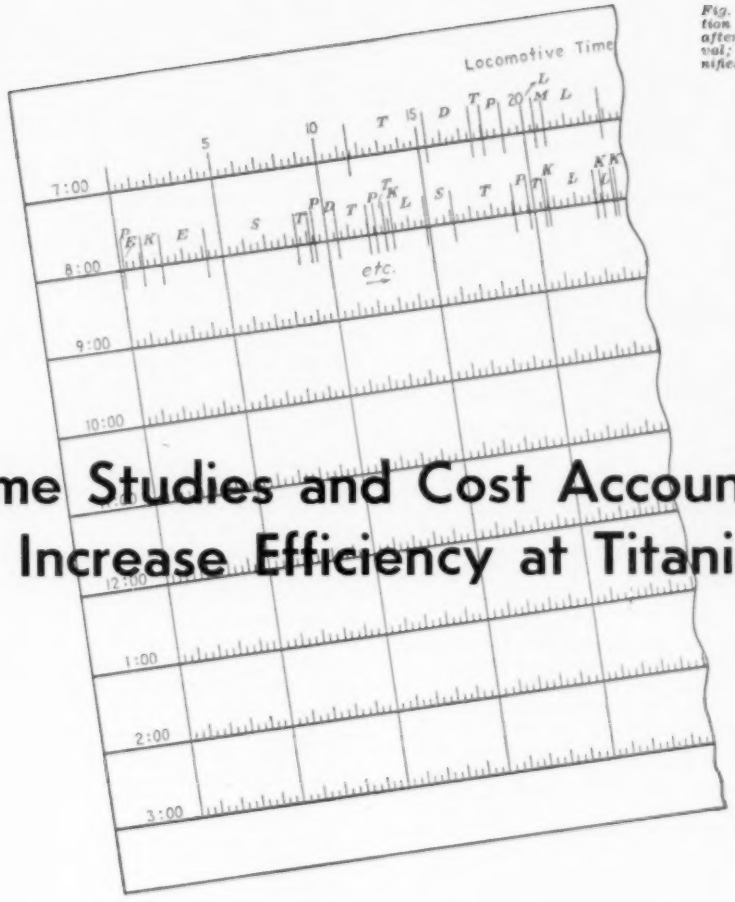


Fig. 1—Cyclical operation recorded by line after each time interval; symbol above signifies operation.

Time Studies and Cost Accounting Increase Efficiency at Titania

by Charles D. Hoyt

Mr. Hoyt, a Junior Member of AIME, recently returned from a tour of Europe on a Fulbright scholarship. During his stay there, he made a study of the Titania mine about which he writes.

TIME studies and cost accounting were combined to increase the efficiency of the overall mining method and thereby reduce costs at the Titania mine of the National Lead Co. in southern Norway about 70 miles southeast of Stavanger. From these studies a more effective drilling system was developed, standards were adopted for setting base pay rates, shrinkage stoping was abandoned for open stoping, a new mechanical loading chute was adopted, and the loading and haulage system was changed.

Ore minerals at the Titania mine are ilmenite and magnetite; gangue minerals are plagioclase and hyperstene with accessory minerals of biotite, pyrite, chalcopyrite, and spinel. The average content of the ore is 17 pct TiO_2 and from 20 to 25 pct iron. The ilmenite crystals contain microscopic lamels of hematite, making it impossible to obtain a high TiO_2 content in the concentrate. An ilmenite concentrate with 44 pct TiO_2 and 36 pct Fe, and a magnetite concen-

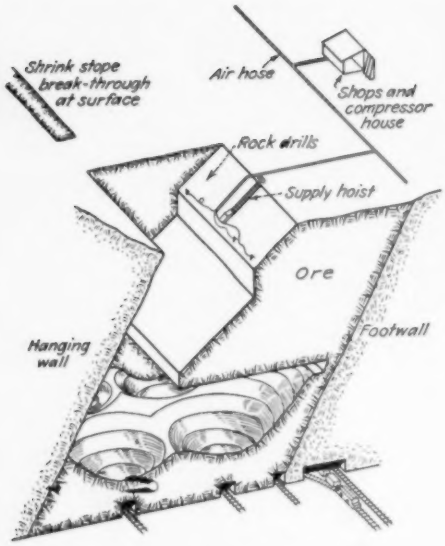


Fig. 2—Benching from the surface proceeding after the shrink stope along the hanging wall has been completed and the undercut by sublevel stoping finished.

trate with 65 to 66 pct Fe and less than 2 pct TiO_2 are made. The magnetite concentrate also contains about 0.65 pct V_2O_5.

The ore body has been deposited in a fracture in anorthosite, dips 45 to 60 pct to the North, strikes ENE, outcrops at the surface over a distance of one mile and has an average thickness of 150 ft. Reserves of 30 million tons of ore are delimited.

The original exploitation of the ore did not start until shortly after the first World War. At this time mining was started with open cuts. Operations were on a small scale and from 1919 to 1935 aproximately 500,000 tons of ore were removed in this manner.

Plans for underground mining on a large scale were made in 1935. During the planning for extended production, open-pit mining was considered but dismissed because for each ton of ore that was to be mined there would have been 2.5 tons of waste rock to handle which would have to be drilled and blasted. Since the hanging wall and the footwall are composed of solid anorthosite, the ultimate mining method selected was transverse shrinkage stoping.

The mine was opened with an adit 90 ft below old workings. Horizontal development commenced with a double-tracked footwall drift and an empty car hanging wall drift; crosscuts were driven through the ore on 45-ft centers. The stopes were 60 ft wide with alternating pillars of 24 and 36 ft. The loading was performed with timbered Chinaman chutes.

The installation of two large compressors gave the mine a total delivery of 3100 cfm at 100 psi. Thirty-six ton storage battery locomotives were purchased and 75 side dump mine cars (70 cu ft capacity). Two loading machines were procured for loading drift rounds.

The ore deposit has many clay filled fissures which caused heavy rock slides from the stopes. Fortunately these all occurred on Sunday or off shift. This danger caused the consideration of new mining methods.

Occupation during the war years provided the opportunity for studying the operations for improved methods by time studies and revised accounting. Mining was continued as usual with one noteworthy exception. Production and efficiency became lower and lower. One glance at the production figures is worth many words in explaining the silent yet totally effective warfare carried on by the Norwegians against Nazi occupation. See Fig. 4, p. 937.

The new method developed is a combination of three types of mining and is applicable only to the upper 200 ft of the ore body which outcrops at the surface, shrinkage stoping, sublevel stoping, and open-pit benches being used.

The ore is removed in 60 to 66-ft transverse stopes. A pilot drift is driven on the strike to determine footwall limits. Later a $7\frac{1}{2}$x$16\frac{1}{2}$-ft double tracked transportation drift is driven in the footwall. Loading drifts are driven on the strike on 60-ft centers. A raise is driven in the center of the stope from the hanging wall drift along the hanging wall to the surface. At the same time a crosscut is excavated from foot to hanging wall 60 ft above the loading drift. From the raise, a hanging wall shrinkage stope is driven to the surface and the ore drawn off. The 60 ft of ore between the haulage level and the sublevel is taken by sublevel stoping as shown in Fig. 3. The next step is to remove ore left between the sublevel and the surface. This work is an open-pit operation and the face is broken down to an angle of about 38°, which is the least slope the ore will run down freely. The ore is broken in 16-ft benches, retreating from the lower portion of the open-pit face. The entire bench is blasted simultaneously with electric detonators. At the same time that this work is in progress, there is a careful scaling of all pillar

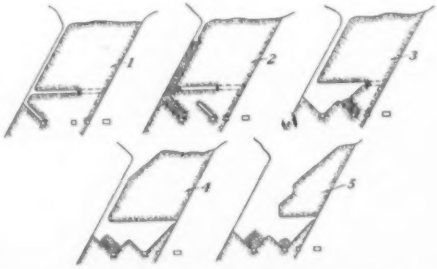


Fig. 3—Drawings 1 through 5 show, respectively, the successive steps in stoping; the hanging wall raise, hanging wall shrink stope, undercutting by sublevel stoping, surface benching begins, stope near completion.

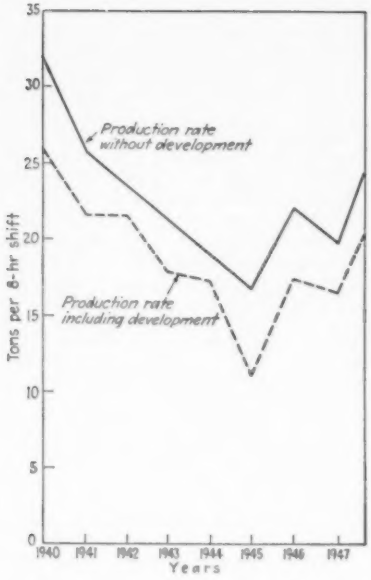
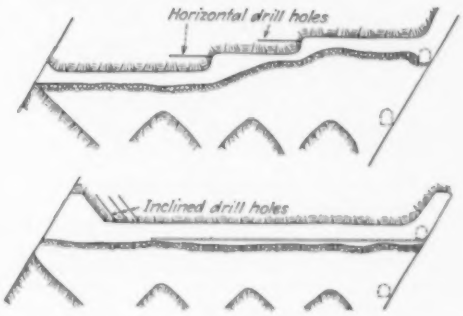


Fig. 4—The plunging curve from 1940 to 1945 indicates the resistance by the Norwegians to the Nazi occupation forces.

Fig. 5—Time-study methods proved the greater efficiency of inclined blast-holes over horizontal ones. Improved fragmentation, reduced roof maintenance, and faster setup time were advantages.



walls to prevent future accidents. The footwall section of the pillar is broken simultaneously with the surface work. See Fig. 2.

The new stoping method has been found to have many advantages over the old system of mining. Ore broken on each shift is loaded and the miners paid on the number of cars taken out. This car tally after each blasting cycle makes it possible to keep accurate records on explosives used per ton of ore broken and therefore excellent control of fragmentation efficiency. Loading ore from open stopes is more efficient because of the absence of ore pressure permitting free flow. Fewer chutes are needed. Loading and drilling can be carried out on the same shift. The new method is safer.

Table I—PRIMARY DRILLING TIME STUDY SUMMARY

	A. Inclined Holes: SAR—85	B. Horizontal Holes: SAR—85	C. Inclined Holes: ft—79
F—	13.9	18.7	13.1
O—	8.8	20.8	5.4
E—	21.5	16.0	18.6
B—	10.2	4.3	8.7
L—	5.1	8.9	6.8
S—	0.3	1.9	0.8
*V—		8.3	
T—	1.1	4.8	0.2
H—	4.6	7.2	2.2
P—		2.7	
A—	3.3		5.9
I—		0.6	

F—lunch, travel-early quit
O—total erection time
E—efficient drilling time
B—changing drill steel
L—tamping and loading
S—blasting time

V—smoke delays
T—scaling the back
H—getting drill steel, explosives
P—blowing out holes with air
A—assistance by helper
I—springing holes

	A	B	C
Drilled feet per shift	57.1	25.9	39.7
No. of drill holes	6	2.3	5
lb. explos. per ft drilled	0.44	0.93	0.44
Efficient drill speed (in. per min)	6.85	4.06	5.31

* No losses due to smoke in inclined holes because there is no springing of the holes as in horizontal drilling.

Electric detonating is used in the inclined holes which greatly speeds the blasting and also gives better results.

Procedure in Time Studies:

To introduce time study work, itemized cost figures of all phases of mining operation are needed. The most costly operations are determined and attempts made to first reduce these. The operation selected must be observed for a few days until the observer is completely familiar with every variation of the work. Next the number of days or weeks during which the study will be conducted must be decided. This is dependent on the individual operation and is a variable factor. The study is then carried out. For convenient recording the Titania mine has developed a tabular form for recording time study data. By placing a line through the appropriate time interval on the chart, the necessary data is recorded. The symbol is placed directly above the time interval. Fig. 1 is a record for a locomotive haulage study. When the studies have been formally completed, they are summarized and presented to the administrative official. Results are thoroughly discussed from every conceivable viewpoint. From the discussions grow the recommendations for changes. Naturally they sometimes verify the effectiveness of the operation being observed.

For several years shrinkage stoping was done with horizontal holes, as shown in Fig. 5, before the question was raised as to whether inclined holes might prove more effective. By a series of time studies of both methods, Table 1, several reasons for the superiority of inclined holes were discovered. With inclined holes the breaking begins at the hanging wall and since less roof area is exposed than with horizontal holes,

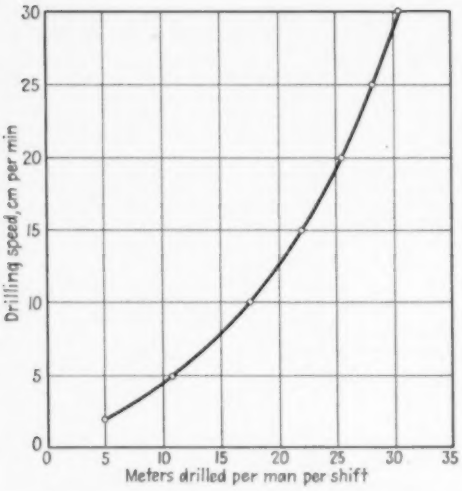


Fig. 6—This curve illustrating the relationship between rock drill speed and distance drilled per man per shift is a standard for comparison of miners and new drilling machines.

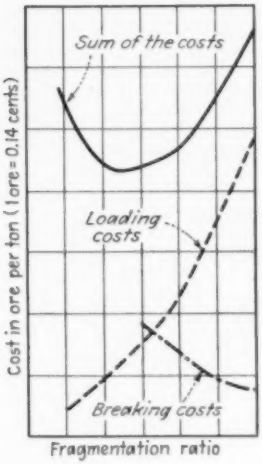


Fig. 7—Diagram right shows point of most economic fragmentation.

a minimum of roof scaling and maintenance work is necessary to make a safe entry from the footwall raise. Another advantage of inclined holes is that the burden is reduced to 3.5 ft, giving sufficient fragmentation to eliminate secondary blasting. Since the retreat is free, all the tools may be left close to the working face. An air pipe may be placed in the stope to reduce the length of air hose used which will give greater pressure at the working face. When working with horizontal holes the length of air hose can be 270 ft at the hanging wall, which greatly reduces the working pressure. When blasting is done with horizontal holes, all the equipment must be removed for each round. The drill set-up time is reduced by the use of inclined holes.

Without a time study and exact analysis of the two methods, it would appear that one method

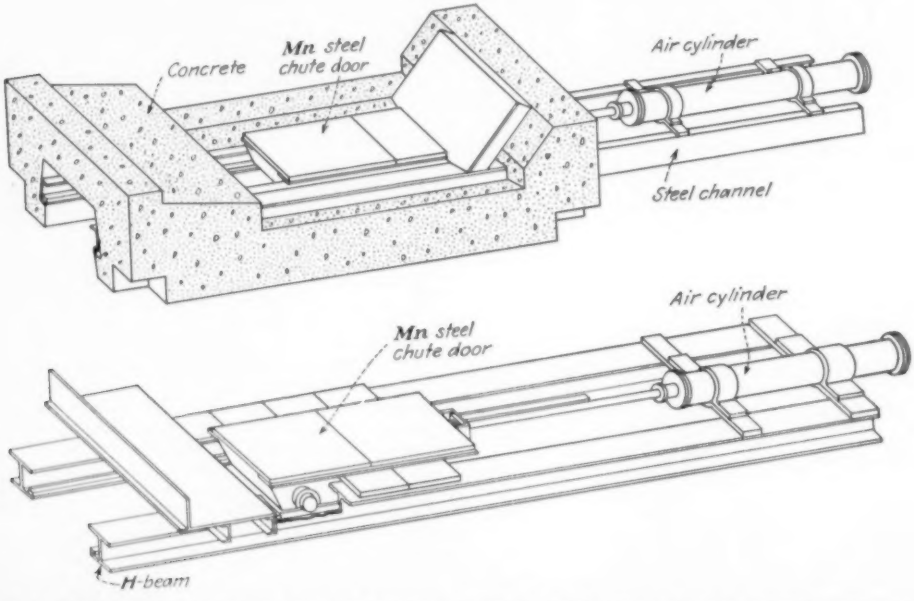


Fig. 8—The efficiency of drawing ore was increased by these chutes. The top one is partially constructed of concrete and therefore permanent, whereas the bottom one can be moved to another draw point as the ore is worked out.

was superior to the other. Such a haphazard perusal may cost time and money, whereas careful time study work and proper interpretation of the results will provide more efficient mining methods at the lowest costs.

Another application for time study methods in the mines is the establishment of standards for various operations. An illustration of this in the Titania mine is shown in the 15-day study of drilling in one of the stopes. From these studies the composite curve was calculated for the various drill speeds versus the drilled meters per man per shift. Thus in similar work there is an accurate basis for calculating the work that could be obtained by a new drill with a higher drilling speed or the evaluation of the various miners' output. This curve is shown in Fig. 6.

One of the first problems was to establish the economic fragmentation of the ore. By accumulating data on the costs of blasting and loading, these values were plotted against the varying fragmentation ratios, as shown in Fig. 7.

The more coarse the fragmentation in the stopes the higher the loading costs and lower the breaking costs. Conversely, fine fragmentation gives high breaking costs and low loading costs. Careful accounting figures and experience will indicate the optimum amount of drilling for each ton of ore.

New Mechanical Loading Chute

The new open stope method presented the need for a more rapid and efficient means of loading at the chutes. Chutes are needed as soon as development begins since there is no delay such as in shrinkage stopes.

Through a series of time studies the idea for a new mechanical chute was developed. The principle of it is merely a conventional Chinaman chute with the planks forming the feeding point replaced by a feeding car which covers the loading slit. The first two models of the loading chute worked quite effectively but each had its disadvantages. The first type was made of wood clad with iron plates. This model resulted in high repair costs. The next version was constructed of reinforced concrete, as shown in Fig. 8. This also operated satisfactorily but it had the disadvantage that it was a permanent structure which made it costly.

The final model, which is in service today, is an all steel construction manufactured in the shop. Two transverse beams are hitched to the drift walls and from these the total construction is supported. Two longitudinal beams form the top and these are topped with a 2½-in. manganese wearing plate which forms the loading slit (7 ft 10 in. long and 28 in. wide). Along the flange of the longitudinal beams the feeding car is run on wearing rails (Fig. 8).

The top plate of the feeding car slides above the wearing plates of the longitudinal beams so as to give a positive feeding action. The feeding car is moved by a 12-in. air cylinder operated from a valve at the platform. The feeding car is topped by two manganese steel plates and is carried on manganese wheels with roller bearing axles.

A vertical cross section of the chute as it appears in the draw point is shown in Fig. 9. A heavy chain screen is hung in front of the operator's platform to protect him from falling rock and a safety chain is provided for his support while working over the chute.

During the stope life approximately 100,000 tons of ore will pass the draw opening neck; this causes a great deal of wear on the rock neck. As it is slowly worn away the ore will flow too far into the loading chute and endanger the operator. To encounter this difficulty the transverse beams are placed in slots in the wall. Then as the rock wears away the entire construction is moved back to compensate for the wearing action.

After the stope is totally worked out the chute may be removed and transferred to the next new loading point. This is done by removing 16 bolts and transporting the entire chute to the desired point.

It is essential that large mine cars be used in the operation because the changing time for a large car and a small car is just the same. Time studies have made it clear that the loading time for an 88 cu ft car and 140 cu ft car is exactly the same. The loading capacity of the chute is

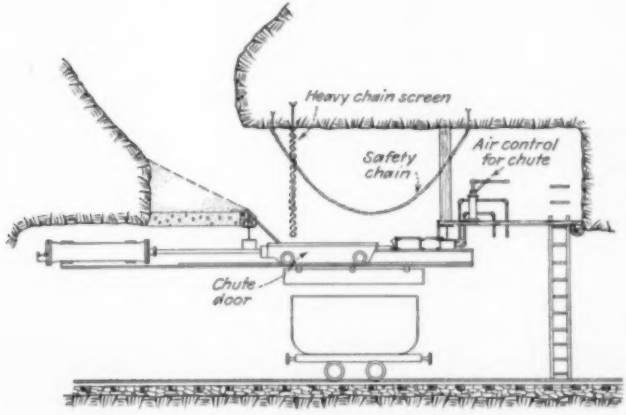


Fig. 9—Cross-section of the chute in place showing control platform, chain screen, and safety chain.

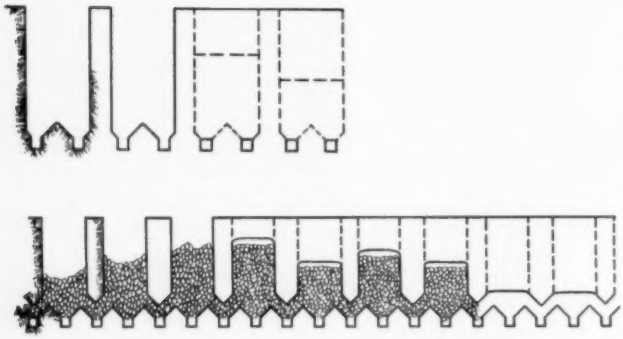


Fig. 10—Open stoping, above: 4 stopes, 16 mechanical loading chutes, working length, 300 ft equal, below: 9 shrink stopes, 72 hand loading chutes, working length, 810 ft equal 400,000 tons per year.

directly in proportion to the size of the mine car, within reasonable limits. Titania is at the present replacing all 88-cu ft car bodies with 140-cu ft ones. This chute has been specifically designed to withstand the enormous wear of loading large quanties of coarse broken ore.

When the first mechanical loading chutes were installed a number of time studies were conducted to determine actual loading efficiency. The figures summarized below are conservative because since these studies, several developments have been made and now the operators are more familiar with the chutes.

Total Cars Avg.	Total Tons Avg. per Shift 8 hrs	Net Wt per car tons	Effect. load time hr
83.3	322	3.83	2.25

The figures refer to coarse broken ore — with fine muck it will have a tremendous loading capacity.

Maintenance costs are currently from 0.4 to 0.6¢ per ton as compared to the 2.1¢ per ton with the timbered chutes and hand loading. The mechanical chutes give a saving of approximately

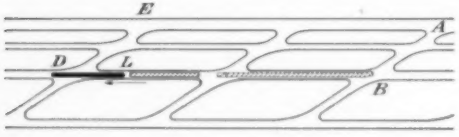


Fig. 11—By changing to longitudinal loading drifts, haulage was speeded.

25 pct in explosive consumption at the loading slit, because large slabs are easily handled.

Titania has ten such chutes in operation and is planning to add ten more when the old wooden loading chutes will be completely replaced.

Mine Transport

Another phase of mining that has been treated by time study work is the haulage system. In the original shrinkage method transverse loading drifts were used. These were not particularly satisfactory for haulage because the crosscuts gave too many switches which caused excessive wear to the rolling equipment, much track repair, and numerous derailments. Studies of the method were made and the results are shown in the table.

HAULAGE STUDY

Symbol		Pct
F	Lunch	15.8
S	Ore Dump Switch	14.7
E	Moving loads	13.5
T	Moving empties	13.5
L	Moving locomotive	13.2
V	Delay load	9.9
K	Coupling	9.3
P	Switching	5.0
D	Miscellaneous	4.6
M	Moving Supplies	0.5
	Total	100.0

These results indicated a need for a new haulage system. The proposed system that is now being installed is longitudinal loading drifts. These are shown in Fig. 11 along with the proposed system of switching. A locomotive arrives at point A with empty cars; it switches to B and travels along the loading drift to the rear of the train being loaded. From the loading point, L, to D the cars have been loaded. The entire set of loads and empties are pushed past the loading point until the first car of the new set is under the loading point. Loading then proceeds as the locomotive switches out the loads and empties at point E; the entire set is taken to A and the empties placed back in the loading drift and the loaded cars hauled to the crusher.

This system should greatly reduce delays in loading and result in a more efficient haulage system.

ACKNOWLEDGMENTS:

I should like to thank the officials of the Titania mine for their kind cooperation in providing me with the material for this report and their hospitality during my two visits. Specifically I wish to thank superintendent Robert Brun for his invaluable assistance.

References:

"Mining Developments at Aktieselskapet Titania," by Robert M. Brun.

Minnesota Granite Poses Tough Drilling Problem

I would like to hear from some operator who is having a hard time getting 1 to 2 fph with 29T churn drills. The 10.5 fph given as the 29T rate at Chino (Open-Pit Forum, ME, May 1949) sounds like easy going to us.

S. J. Harrison

Chief Engineer, J. L. Shiely Co.
St. Paul, Minnesota

One of the operations of the J. L. Shiely Co. is quarrying in a hard granite gneiss with intrusions of gabbro or trap. During the winter of 1948-1949 the quarry ramp was lowered about 30 ft and during the 1949 season practically all drilling was done from the quarry floor, developing a face from 72 to 83 ft for the entire quarry area. The performance and cost table shows our record for 1948 and 1949.

Because the rock from the quarry is crushed and sized we do not hesitate to spend a little more for explosives or secondary drilling if so doing will profitably increase the shovel output to the plant. The secondary drilling cost was reduced in 1949 by the use of Kennametal bits as shown in the table. Thought has been given to the use of a drop ball rig for secondary breaking but the 2-acre area below ground level now being worked scarcely offers opportunity for experimenting. Such a rig would have to operate for less than $9.40 per hr, including depreciation, to equal the present ton cost of $0.0564.

We have been increasing the weight of tools used with 29T drills. In 1949 we increased the weight from 2650 lb being used in 1948 to 3000 lb. The weight has been further increased to 3500 lb for 1950. Beyond this it seems that a new device is necessary to improve footage in this hard granite.

Performance for Quarry Breaking with 29T, Drilling 9-in. Blast Holes

Unit	Total	1948	1949
Ft drilled	8728	3606	5122
Hr drilled	9173	4518	4655
Man-hr	18346	9036	9310
Tons produced	420088	199700	220388
[1] Tons per drill-ft	48¼	55	43
[2] Ft per hr	.95	.80	1.11
[3] Dollars per drill ft, labor and material		4.19	2.97
Dollars per hr, labor and material		3.35	3.26
Dollars per man-hr, labor only		1.24	1.20
[3] Dollars per drill-ft, bit servicing and shop, labor and material		2.36	1.62
[4] Lb, primary explosives		65382	73716
Tons per lb		3.05	2.99
[4] Tons per lb, secondary explosives		17.87	16.52

Operations at the Shiely-Petters Crushed Stone Co., St. Cloud, Minn.

Costs (dollars per ton) of Quarry Breaking with 29T, 9-in. Blast Holes

	Labor		Materials		Totals	
	1948	1949	1948	1949	1948	1949 2-yr avg
Drilling	0.0561	0.0506	0.0196	0.0183	0.0757	0.0689 0.0721
Bit servicing and shop	0.0230	0.0273	0.0196	0.0105	0.0426	0.0378 0.0401
Primary explosives	0.0030	0.0035	0.0556	0.0577	0.0586	0.0612 0.0600
Secondary explosives	0.0025	0.0024	0.0095	0.0104	0.0120	0.0128 0.0125
[5] Secondary drilling	0.0376	0.0265	0.0315	0.0171	0.0691	0.0436 0.0558
Quarry breaking costs	0.1222	0.1103	0.1358	0.1140	0.2580	0.2243 0.2405
Shipping, pumping, etc.						0.0173 0.0085
Total						0.2416 0.2490

[1] Tons per ft of 9 in. hole drilled in 1949 decreased because the drilling for lowering the quarry floor was spaced 24x27 ft instead of the usual 27x30 ft for face drilling 1948.

[2] The feet drilled per hr increased 40 pct due, no doubt, in 1949 to easier moving conditions on quarry floor and improvement due to experience. Nickel bits gave little extra footage per se, but did increase change time interval about 50 pct. Added weight of drill tools to total of 3000 lb in 1949, increased footage also, probably about 20 pct.

[3] The nickel steel bits made some saving by decreasing the number of bits required per hr, thus reducing overtime in shop. Improvement in blacksmith's technique from experience also helped.

Total drill and bit cost per ft, $6.55 for 1948; $4.59 for 1949.

[4] Explosive prices increased ¼¢ per lb in 1949.

[5] Decrease in secondary drilling costs due largely to Kennametal bits.

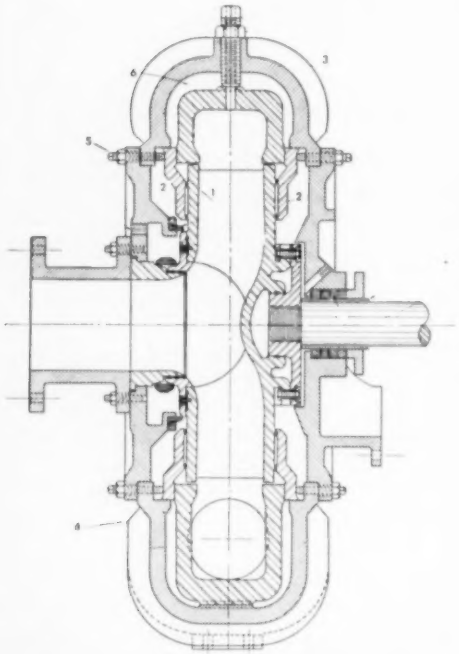
Dredge Pump Life Increased by

By George T. Bator

If confronted with the problem of pumping a mixture of slime-free sand, gravel and boulders up to six in. in diameter, at the rate of 175 tons per hr in one single-stage pump against a static head of 90 ft and a total head of 140 ft, of which 20 is on the suction side, an engineer might be forgiven for backing away from

Mr. Bator is in the mining department, Colorado School of Mines and a member of AIME.

the job, especially if he were also told that the pump wearing parts must have a minimum hardness of 550 Brinell for the sake of abrasion resistance. Yet precisely that feat has been accomplished in the gravel pit of John B. LaGarde, Inc., near Anniston, Ala. The pump used is of decidedly novel design and its successful operation since installation 1½ years ago suggests that it may be of timely interest in numerous other applications in the mining industry, such as pump transportation of coal washer refuse or rock from open-pits without unduly fine crushing.



Cutaway view of the Thomas Pump shows: (1) impeller, (2) side liners, (3) casing, top half, (4) casing, bottom half, (5) jack screw and nut, (6) water jacket.

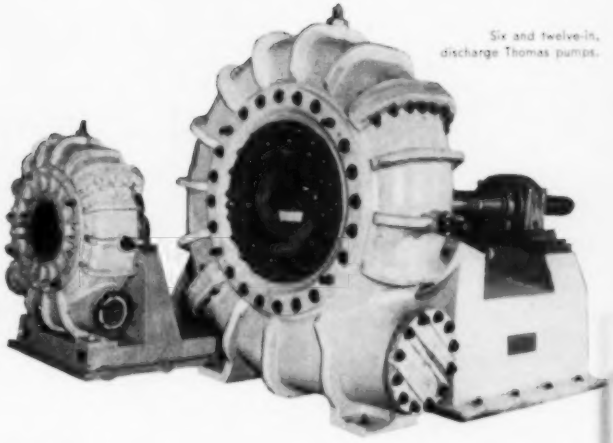
The terrain surrounding the LaGarde pit is relatively flat and when the pit was first opened up some years ago, a pump of semihard gray iron was satisfactory for the relatively light duty at that time. As the pit deepened and boulders began to be encountered, it was necessary to change to manganese steel for the sake of both abrasion and shock resistance. When the floor of the pit reached 90 ft below surface, it became obvious that the limit of single-stage work for the pump in use had been reached and a second similar pump was purchased for a booster.

In the meantime, Thomas Foundries, of Birmingham, had undertaken the design of a dredge pump that could be made of Ni-Hard. The superior abrasion resistance of Ni-Hard is generally recognized but it is admittedly attained at the expense of decreased ductility so that the use of Ni-Hard had not been feasible in large dredge pumps of conventional design.

In the pump now to be described, this difficulty was circumvented by taking advantage of the incompressibility of water. As shown in the drawing, a one-piece Ni-Hard casing liner of volute design is mounted in and surrounded by an outer casing of ordinary cast iron. Within this casing, two Ni-Hard side liners are held in place in a smooth fit against the casing liner by means of jack-screws through the outer side plate. These, together with a top jack-screw, position the assembly and the Ni-Hard impeller operates within the enclosure so formed. The construction and the relation of the shell liner to the outer casing is such that there is a space of approximately 1-in. between the two which is filled at all times with clear water, so that the shock and impact caused by heavy boulders, rams, water hammer or blow-backs, are cushioned by the confined body of water and thus transmitted to the heavy cast-iron outer shell. It is this feature of the design that enables the extremely hard and nonductile Ni-Hard to withstand the heavy shock loads. This putting of the gland water to double use, so to speak, also insures that no damage can be done to the outer casing should the casing liner wear through.

The first pump manufactured in this design was installed in the LaGarde gravel pit in Sep-

- hard alloy
- water cushion
- double casing



Six and twelve-in. discharge Thomas pumps.

tember 1948 and has operated successfully in a single lift since that time. The pump is driven by a 200-hp Diesel engine at approximately 600 rpm through a V-belt drive; it has a 10-in. suction, 8-in. discharge, and 30-in. impeller. The centerline of the pump is approximately 15 ft above the surface of the water in the sump and there is 30 ft of 10-in. pipe. The total static lift is 90 ft from sump water level to the point of discharge, equivalent to a total head of 140 ft.

In operation, the sand and gravel is sluiced to the pump intake by means of hydraulic monitors and the feed is consequently irregular in both size and rate. On the average, the pump is moving 2700 gpm and the output averages 150 to 175 tons of sand and gravel per hr. The space between impeller shrouds is 6½ in., so that rock of sizes approaching the capacity of an 8-in. pipe occasionally pass through and do so without difficulty. They do not require extra push water because they pass through the line piston-fashion. Occasionally a larger piece will come up the 10-in. suction line. The only evidence the operator has at such times is the laboring of the Diesel and he takes care of the situation by cutting off the engine momentarily to permit run-back.

Looking down discharge line to shack housing Thomas pump at bottom of J. B. LaGarde, Inc. gravel pit near Anniston, Ala. Booster pump had been planned, but was never necessary.

From the operating viewpoint, the fact that the installation of a booster pump was rendered unnecessary is indicative of satisfactory power utilization. The life of the wearing parts in the manganese pump previously used ranged from two to a maximum of four weeks, depending on the character of material currently being handled. The Ni-Hard wearing parts of the new pump have a minimum hardness of 550 Brinell and a life from 5 to 7 times longer than manganese steel parts.

A second pump, identical with the one above described, has been installed at the pit of the American Sand & Gravel Co. Hattiesburg, Mississippi. This pump, also Diesel driven, is working against a total head of 110 ft, handling 175 tons per hr of gravel up to 1½ in. in size, from which the —¼-in. fines have been screened. More recent installations of the pump have been made.

Altogether, the fundamental novelty of these pumps is that they are of a hard alloy in a service where some ductility of metal has always been regarded as essential. The unconventional design features are simply those required to take advantage of hardness without penalty from lack of ductility.

Mine and Surface Maps

By Neil O'Donnell

MODERN mine maps are largely a product of evolution. The first mine maps used in the west were old composite maps with all the levels plotted on the same sheet. The composite was widely used and occasionally is seen even now. This type of underground map had serious limitations. One was the development of an indecipherable complex of workings around steep shafts. The other limitation was that of size; often when a working passed the limit of the sheet no map was made of it.

These old maps have been superseded by the single level maps, laid out in squares or rectangles which can be expanded indefinitely in all directions. This produces a system capable of unlimited expansion in three dimensions, with all the detail of every working clearly marked and legible.

Mr. O'Donnell is executive vice-president and general manager, Idaho Maryland Mines Corp., Grass Valley, Calif., and an AIME member.

A good set of mine maps should be laid out in blocks, all blocks of equal size. Each block should be limited by plane coordinates without overlap on bounding map quadrangles. At all times the direction of North should be indicated as an aid in remembering bearings of underground workings.

It is desirable that all maps of whatever scale be plotted on sheets of uniform size. In fact, it is possible to make nearly all underground maps, surface maps and sections on sheets of the same size.

Plane Coordinates

Most metal mining operators employ a system of plane coordinates for location of mine and surface workings. The coordinate system should be devised so all of the mapped area is in one quadrant, preferably the northeast quadrant. Contrary to belief of some the 0.0 point of the coordinates need not be a known point. For example, the 0.0 point of the Los Angeles city coordinate system is in the Pacific Ocean.

Surface Maps

Most companies require information on topography, geology, property lines, assays, culture and triangulation nets. For large areas, scales less than 200 ft to the in. are not recommended if detail is to be mapped. At Idaho Maryland the surface maps are on 200 ft scale. Each map covers an area 6000x8000 ft, or 30x40 in. inside coordinates. Map blocks are designated by letters reading from west to east and by numbers reading from south to north. On this scale one set of letters will cover expansion to a width of nearly 40 miles; the possible expansion north and south is nearly limitless.

The reason for numbering from west to east and south to north is that all workings are in the northeast quadrant and addition of letters or numbers are in the direction of possible expansion.

The base map for all surface surveys is the hard copy. Hard copy sheets are usually a good grade of mounted paper with a cloth back. From the hard copy tracings can be made for a wide variety of purposes. The types of tracing used are dictated by the problems management faces. In some places a single tracing of each hard copy has everything on it, including geology. As a general rule, however, geology is not plotted on hard copy maps and separate tracings are best for geology. Geological tracings should have contours, roads, streams, diamond drill holes, culture, shafts, dumps, trenches, and cuts. They may or may not have mining claim or property lines. In some places these are desirable on geo-

logical maps, in other places they are not. Pipe lines and power lines may be useful for location purposes, otherwise they should be left off the sheet. The fewer the non-essentials on a map the better. Geological tracings should have the coordinate system ruled with spacing of coordinates uniform in both directions. This spacing may vary from 100 to 1000 ft. It is doubtful if any interval would be desirable at less than 100 ft, or more than 1000 ft. This map should also show traces of sections in use for determination of the surface outline on sections and for ready transfer of surface geology to sections.

In some instances assay values determined from trenches, cuts or holes can be placed on surface geologic maps and it makes a desirable combination; at other times it is not desirable and assays better be placed on a separate sheet. If necessary, both of these sheets may be placed over a shadow table in register and the relative position of all points noted. The decision to use one sheet for geology and assays or separate sheets for each is a decision to be made locally.

Claim ownership or property boundaries may also be placed on the geological map and often are. Where the claim set-up is complex or where there are extensive grants of rights-of-way for power lines, ditches and roads or surface property ownership, which is not coincidental with mineral land ownership, a separate tracing for properties, rights-of-way and other delineation of rights or ownership is most desirable. This map may be used in conjunction with other tracings over a shadow table to determine relative positions of points such as relation of geological information to property boundaries.

The third desirable tracing of surface workings is the culture map. On this map should be placed topography, roads, streams, bridges, reservoirs, buildings, tramways, shafts, tunnel portals, power lines, pipe lines and anything else that a construction man might need for new construction or location and maintenance of old construction.

The scales most used on surface maps are 40-50-60-100-200-400-500-600-800 and 1000 ft per in. Of these, the multiples of 50 ft are best. Only 50, 100 and 200 ft per in. are recommended for working maps and 400, 500, 800 and 1000 ft per in. for maps requiring larger scope and less detail. The scale in this case would be determined by the objective of the map.

For metric scale maps there is a general drift to multiples of 500 rather than 50, that is: 1:500, 1:1000, 1:2000, 1:5000, 1:10,000. The 1:500 scale more nearly approximates a scale of 1 in. to 40 ft (being 1 in. = 41.666 ft) than the more widely accepted 1 in. = 50-ft scale. Metric scales have decided advantages over the in.-ft type of measurement. It will also obviate conversion of maps from inch scale to metric scales for legal purposes—a job which can become very tedious.

Underground Maps

For underground maps a scale of 50 ft per in. is recommended. It is large enough to get a great deal of detail in geological mapping and it is small enough to get a block 1500x2000 ft on a 30x40 in. sheet. At Idaho Maryland each 20 ft scale surface map covers sixteen 50-ft scale underground maps. Sheets for underground maps are also 30x40 in. inside coordinates. This size is recommended because it is easy to handle, files in standard drawers and is easy to work on because points to be plotted on any part of the map are easily reached and tracings are easily printed on standard printing machines. Underground maps are also sub-lettered from west to east in each 200-ft scale block and from south to north.

A hard copy base map is recommended for underground maps. These sheets should have only workings of a single level on them, together with survey stations, survey station numbers, diamond drill holes, diamond drill hole numbers and elevations where desired. These maps should have coordinate lines in some color other than black and should also have traces of sections, not on coordinate lines, in some color different from either the coordinate color or black. Elevations should be given on enough points so that geological sections can be made which show the actual elevation of workings, rather than presumed elevations for workings. This helps materially in working out detailed structure on sections.

Diamond drill holes may be designated by a color other than black and solid lines used for flat holes and a dashed line for inclined holes. If holes are not surveyed the presumed length and bearing should be plotted. If they have been surveyed, the actual bearings and distances should be used. On inclined holes only the horizontal projection is desired. Black ink is best for workings and survey stations as well as all lettering. It is easiest to see through tracing cloth.

The tracings recommended for use in ordinary mine procedure are: (1) a geological map, (2) an assay map, and (3) a stope map. In some places and under certain conditions the geologic map and the assay map are the same, but as a general rule, the assays should be plotted on a separate map. The practice at some properties, where gold or silver is mined, of using the dollar value of these metals on assay maps is a bad one. Definitely the oz per ton or g per ton should be used, otherwise price fluctuations render old maps inaccurate and inefficient. Assay information should show in order: (1) the width or length of sample channel, (2) percentage of the most important metal, (3) percentage of the next most important metal, etc. Ounces per ton or g per ton can be substituted for percentage. In vein type deposits location of the sample is desirable because more than one sample may cut across a face. For example, the first sample may

be the hanging wall, second the vein, third the footwall as exposed.

In some places vertical sections are more desirous for plotting assay information than plan maps. At Potosi, Bolivia, it was found that sharp outlines of stopeable ground could be determined by putting assays in drifts on vertical longitudinal sections of each vein. This was in a type of vein in which it was impossible to tell by visual examination whether the vein contained tin or not. This method of plotting assay information may have a wider use than has been generally believed. In this particular instance it was remarkable how well the ore shoots could be outlined on these vertical sections.

Combinations of plan maps and vertical sections for plotting of assay information have been made with the idea of using the maps for several purposes. At Huanchaca, in Bolivia, there was a long vein which was essentially a single fissure. Assay maps were made which had, at the top, the drift on a level with its assay information; at the bottom was the next drift below with its assay information; both in plan as taken from the hard copies. Between these two was a long section which showed the stoped and unstoped ground between these levels. This mine produced, in order of value, silver, lead, zinc, copper and gold. Fluctuations in value of one of these elements often determined whether mining was to cease, if a stope was operating, or start if not operating. If operating and it was found necessary to cease operation, samples were cut in the back of the stope and plotted on the longitudinal section, available for the next fluctuation of values.

These maps were used for ore computations also, prints being made annually and filed with ore computations. The mine superintendent and his staff found this type map most useful in determining, quickly, tonnages and grades of ore available in unstoped areas.

The third type of tracing usually required at a mine is the stope map. Of these there are varieties almost without end. The simplest for a single fissure mine with a vertical vein is a vertical section in the plane of the vein on which raises and stopes can be recorded. In mining flat lying deposits the plan map is the stope map. From this simple beginning an almost limitless variety of combinations can be evolved. Where the vein system becomes very complex, or large irregular bodies are mined, outlines on regular or stated intervals must be resorted to which are worked up into books, one sheet above the other, with an outline of the stoped area on each sheet at its respective level. Most small to medium mines fall in the zone between the simplest and the complex, although they may have veins which require any one of the above treatments. No one of these methods is ideal. Most orebodies are neither vertical nor horizontal, nor do they usually maintain uniformity of dip or outline. Some properties make use of vertical long sections for veins whose dip is greater than 45° and horizontal projections where the dip is less than 45°. In both of the above cases the projection gives a relatively close approximation of the length of the block but the length up or down the dip is foreshortened and is always less than the actual dip measurement. Others prefer the "plane of vein section" in which the length of the ore body on the sheet and its dip measurement are reasonably close to the actual length and dip measurement. The latter has the advantage that computations of ore tonnage can be made from the map more easily than in the case of the first two types. Either of the above stope maps requires less work in measurement and plotting than is the case where outlines must be measured on every floor of square set stopes or at more or less regular vertical intervals of excavation. However, as mentioned previously, for certain types of deposit no other method is satisfactory. This latter method has a decided advantage if detailed information is required on the shape of stoped areas for construction of mine models, especially for those models required in courts, or the computation of excavations, because this type takes into account area and thickness, whereas the other types take into account area only, unless marked with vein thicknesses. With the above limitations and weaknesses in mind, choice can be made of the type most desirable for the immediate deposit to be mapped.

Vertical Sections

Vertical sections at regular intervals throughout the working area of vein mines are essential. These sections give the third dimension to a map system and are extremely valuable for correlation of geology from level to level but they also serve as a check on mapping of geology on the levels. Often a projection of a structure on a section will not coincide with mapping on the level with the result that the level mapping will require checking. Very often the section will be found correct and the level mapping wrong.

Sections should show all workings and diamond drill holes which cut the plane of the section, in solid lines. It should also show adjacent workings projected usually no more than 50 ft, in dashed lines. Drifts should be plotted at elevations where they actually exist, otherwise the task of making geology fit the workings will be found to be a rather difficult one.

Scales used on sections are usually smaller than those used on the plan maps, although there is no reason why they cannot be the same. In all cases the vertical sections should have a scale which would allow having a complete section from top to bottom of the mine on a single piece of tracing cloth. Those in use at Idaho Maryland are 100 ft to an in.

Air Compressors for Roof-Bolting Work Introduced by Davey Co.

A new line of mine air compressors for handling all types of roof-bolting work has been announced by the Davey Compressor Co., Kent, Ohio. Outstanding feature of the compressors is their low height, the four standard models measuring 32, 33, and 37 in. Units are available in four standard two-stage sizes to deliver 105, 160, 210 and 315 cfm at 100 lb pressure.

Equipped with pneumatic tires and automotive steering, they are capable of 30° turns, and can reach many places inaccessible to ordinary mine compressors. Flanged steel wheels are available as optional equipment. All working parts are enclosed in a metal housing. Other features include Davey peak efficiency valves, oil pressure safety switch, full force feed lubrication and built-in unloader system. (Circle No. 1)

☆

Froth-Flotation Separation Unit for Coal Preparation

A froth-flotation separation unit has been added to the Wilmot Engineering Co. line of coal preparation equipment. Pilot plant and field tests show that it has increased yield of marketable fines by 35 pct. All operations have been concentrated in one unit, with resulting economy in floor space, operating time, power and labor costs. The new unit is completely automatic. Cone diameters in the 4 standard models are 6, 9, 12 and 16 ft, recovering fines of −3/64 to +100-mesh size. (Circle No. 6)

☆

Circular Bin Discharger Offered

The S-A Circular Bin Discharger, designed for handling pulverized, granular, glutenous and soft lump materials that may refuse to discharge by gravity, is described in a new bulletin issued by the Stephens-Adamson Mfg. Co. Two types are available, one with flight fingers above the bin bottom, the other with flights above and below the bin bottom. They carry material to discharge openings and give a measured flow in the case of two-stage units. (Circle No. 4)

Manual Jet-Piercing Blowpipe

A jet-piercing blowpipe that can be held in the operator's hands and used for piercing blastholes in rock quarries is a new product of the Linde Air Products Co. It is claimed that holes of $1\frac{1}{8}$ in. diam can be pierced to a depth of 5 ft in hard rocks and ores, such as sandstone, quartzite and granite. The pipe can be used at any angle for vertical or horizontal holes, as well as diagonal holes pierced from below in protruding rock. Larger blowpipes can be supplied for piercing holes to a depth of 7 to 9 ft or more. The pipe burns a mixture of oxygen and either kerosene or fuel oil. (Circle No. 3)

☆

Two New Types of Plastic Pipe Offered

Carter Products Corp. has announced two new pipes in the CARLON line, called Carlon EX and Carlon L. The EX pipe is flexible, lightweight, and said to be absolutely rot, rust, and electrolytic resistant. It is suitable for use as an air duct, or in exhaust systems or for electrical conduit work. It can be used with suction pumps to transmit sludge or expel acid waters from mines. It is furnished in standard sizes, weighs only 61 lb in 100-ft lengths 2 in. in diam.

The "L" pipe is a rigid plastic pipe, developed from a co-polymer base, and recommended for use in industry mines as an air line or for transmission of liquids, vapors and gases under moderately high pressure. High tensile and flexural strengths are claimed, and burst pressure is said to be 900 lb for a $2\frac{1}{2}$ in. diam section of the pipe. (Circle No. 2)

☆

Mighty Midget Hand Hoist

The Mighty Midget, a hand-operated hoist or puller, weighing $9\frac{1}{2}$ lb, taking up less than 1 ft of space, and capable of lifting a full 1000 lb load, is being offered by the Coffing Hoist Co. It features a double-purpose handle which can be used as a straight lever for heavy loads, or doubled over at right angles to itself to form a crank for high-speed lifting or pulling. The 100 lb capacity model sells for $27.50, and a 500 lb model is priced at $19.75. (Circle No. 8)

News Flashes

• A new machine which examines grease samples under stroboscopic light has been developed by the Texas Co. to determine changes in quality, texture, consistency, as well as tendencies toward aeration, leakage and breakdown through heat and prolonged use.

• New territory near Carlsbad, N. Mex. is being tested for potash. Diamond drills are taking test core at 1500 and 1600 ft, and a mixture of salt and potash is being found at 900 to 1500 ft in layers to 12 in. in diam.

• The biggest current news in mine safety is the Mine Safety Appliance Co.'s John T. Ryan Memorial Laboratory in Pittsburgh. Its facilities are available to all industry.

• A new Mineral Testing Laboratory has been opened in San Francisco by the Western Machinery Co. It serves metallic and nonmetallic mining industries, coal preparation, sand and gravel and other related industries. A new Mineral Testing Department has been formed to run the labs.

• Large-scale production of gas turbines has been started by the General Electric Co. in Schenectady, N. Y. More than 20 gas turbines are scheduled to be built within the next to years for power generation and for gas pipe line pumping stations.

• Carbon brushes can now be obtained at lower prices because of a new merchandising scheme developed by National Carbon. Quantity purchasing of "standard brushes" packaged for easy storage and shipment makes this possible.

New Ultraviolet Light Source

A new ultraviolet light source with inbuilt dark chamber, for fluorescence detection and comparison in daylight is being offered by Menlo Research Laboratory. Known as the Fluoretor, the device operates on standard flashlight batteries, and includes six various specimen-holding caps for examining various substances. It has a swivel head for left or right hand operation, and it can be converted from a daylight-viewing unit to an ultraviolet flashlight for scanning large areas. Sells for $49.50, or $35 without case and special eyepiece attachment. (Circle No. 5)

Trolley Tractor for Mine Haulage

A trolley tractor to provide a supplementary transportation system for mines using belt conveyors for main line haulage is announced by the Baker-Raulang Co. It is arranged for trolley or battery operation.

No charging station is required, as in the main entry the tractor takes power from trolley wires, and beyond the trolley system the battery provides power for moving about working areas.

Advantages claimed are: safe for hazardous areas, speed for long hauls, no charging equipment required, combines speed of trolley operation with trackless flexibility of battery power. (Circle No. 7)

Manufacturers' Bulletins

For your convenience, a listing of booklets and other material currently being offered by the manufacturers. To obtain this information, merely circle the desired number on the coupon, and return it to MINING ENGINEERING.

1) RUST PREVENTION: A new catalog issued by *Rust-Oleum Corp.* offers solutions to many industrial rust problems. There are at least 70 color chips designated and reproduced. Resistant qualities, drying time, thinner requirements and methods of application are completely described.

2) DIESEL TRAMMER: A bulletin describing a 2-ton diesel mine trammer is offered by the *Ruth Co.* Power line, clutch, gears, axles, bearings, tires, side-rod drive, brakes, and exhaust gas conditioning are all individually described. Various sections of the trammer are illustrated by close-ups.

3) SYNCHRONOUS MOTORS: Several articles citing the various uses of these motors are included along with illustrations of motors and generators in catalog 200-SYN-31 available from *Electric Machinery Mfg. Co.* A synchronous motor performance chart is shown with directions for finding power component of current to motor, hp output of motor, operating power factor of motor, and reactive component of current to motor.

4) ROLLER CHAINS: A complete packaging program for roller chains and parts has been announced by the *Morse Chain Co., Div. Borg-Warner Corp.* Bulletin F 57-50 gives complete details with all available standard packaged quantities and package weights and list prices.

5) TRANSFORMERS: Reactors and filters are also included in 22-page booklet available from *United Transformer Co.* Various models are illustrated along with dimensions, weight, voltage and current. A price list on the complete line is given.

6) STEEL SELECTION: A 4-page brochure lists all popular grades of hot-rolled and cold-finished carbon and alloy steel bars, giving principal characteristics, mechanical properties and uses. Issued by *Joseph T. Ryerson & Son, Inc.*, it is intended to serve as a convenient guide to the selection of bar steel for various jobs that come up in the average shop.

7) SERVICE PUMPS: Sketches showing vertical turbine pumps applied to river and relift service, boosting, recirculation, cooling tower, gas and oil pumping, and drainage are contained in 16-page bulletin offered by *Layne & Bowler, Inc.* Cut-away drawings of pump bowls and discharge column are also included.

8) POWER SHOVELS: *Thew Shovel Co.* presents its new model in the line of power shovels and cranes in a new catalog. It is the first machine in the 1 yd class equipped with a hydraulic coupling as standard equipment. This prevents stalling of engine due to any digging conditions, increases output, and reduces maintenance by acting as a shock-absorber to keep impact shocks and digging stresses from being transferred to the mechanism or cables and reduces vibration.

9) DYE PENETRANT: This method of inspection for every kind of metal is described in a new circular issued by *Dy-Chek Co., Div. Northrop Air-*

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craft, Inc. Dy-Chek is used on all metals such as nickel, steel, copper, brass, cobalt, aluminum, titanium, magnesium on various parts where surface testing is required.

10) GAS DETECTOR: Made especially for use in plants where inflammable solvents or gases are present this instrument registers the air-vapor ratio at all times. A brochure distributed by *Lor-Ann Instrument Co.* describes and illustrates it. A table is given partially listing gases that are used industrially. It can be made to shut off spark giving apparatus, start emergency ventilation and open valves of neutralizing gas.

11) CARBIDE BITS: A 20-page booklet offered by *Rock Bit Sales & Service Co.* shows how to successfully recondition tungsten-carbide bits. It contains helpful operating suggestions for drillers on how to obtain maximum speed and footage out of carbide bits, and illustrates the correct way to grind chisel bits and cross bits.

12) SPEED REDUCERS: Bulletin G-WBV describes how IMO-De Laval worm gears are manufactured. Available from *De Laval Steam Turbine Co.* it lists types of speed reducers, selection tables, hp ratings, and dimension tables. Diagrammatic sketches illustrate the various models available.

13) TURBINE PUMPS: Construction details, various types of drive and typical installations of deep well turbine pumps manufactured by the *Deming Co.* are illustrated in bulletin 4700. They are used in irrigation systems, mine dewatering, sump pumping service, and loading tank cars. These pumps are furnished for wells with ID of 4 in. or larger. Capacities vary from 12 to 3000 gpm.

14) RESEARCH & TESTING: The basic industries' commercial research and testing facilities used in fields or minerals' beneficiation and communition, oil extraction, and others are described in a new brochure released by *Allis-Chalmers Mfg. Co.* It carries a cross-section of laboratory investigations conducted and of industries served.

15) CONVEYING SYSTEMS: *Convair Corp.* has issued booklet 103 describing its line of pneumatic conveying systems for use in all industries. These systems handle materials ranging from 800 mesh to lumps up to 8 in. in size and up to 350 lb per cu ft in weight. Diagrams illustrate the setups for various firms that use the systems.

16) EARTHMOVERS: A new illustrated catalog has been published by the *Warner & Swasey Co.* covering the recently introduced model Gradall earthmovers. The catalog describes the machine's operation, lists specifications and illustrates it at work on a variety of jobs.

17) JAW CRUSHERS: The advanced design characteristics of *Traylor Engineering & Mfg. Co.* crusher jaws result in greater crushing efficiency. Bulletin 1123 describes the many advantages of using crushers with curved jaws. The frame of Meehanite metal is of box section design and provides maximum strength per lb.

18) BLASTING CAPS: A 12-page booklet issued by *Hercules Powder Co.* gives all the details for using short delay No-Vent caps. They are used in underground mines, open-pit mines, coal stripping and construction, to mention just a few.

19) ROCKER SHOVEL: A brochure in color describes *Eimco's* model 104 Rocker-Shovel for use in sand and gravel. It is efficient for bulldozing sand, carrying and loading rock, gravel and other bulk materials. The model is available with diesel engine or electric motive power. It can be used to carry the load directly to the conveyor hopper, or trucks can be used.

20) PROCESS PUMPS: *Worthington Pump & Machinery Corp.* has published catalog W-341-B14 describing type HM process pumps. Sectional drawings of the vertically-split, single-stage, volute centrifugal process pump are shown. It is capable of pressures up to 450 psi and will withstand temperatures up to 500°F.

21) EXCAVATORS: Over 65 years of designing and building materials handling equipment is behind the model 255-A described in bulletin X71-3 available from *Harnischfeger Corp.* It is possible to make conversions right in the field for service as shovel, dragline, clamshell, crane or trench hoe. The model is available in six types of service.

22) HOSE COUPLINGS: *B. F. Goodrich Co.* has published a 12-page catalog section on hose couplings and fittings of various types. Details and pictures, specifications, general and maximum pressure recommendations, and a description of threads are included.

23) REFRACTORY CONCRETE: A revised edition of the booklet containing the latest information on refractory and heat-resistant concrete may be obtained from *Lumnite Div., Universal Atlas Cement Co.* Coal bins and hoppers, kiln, kiln car tops, stacks, and chimneys are but a few of the industrial applications for this concrete.

24) AIR DRILL: Because of the small size and light weight of this drill, it may be set up quickly and drilling started as soon as the motor is connected to the air line. Bulletin sheet 80 briefly describes this *E. J. Longyear* drill.

25) CHAINS: A booklet containing well over 200 pages is offered by *Wilmot Engineering Co.* describing rivetless chains, conveyors, elevators, pinions, sprockets and many other attachments. Bulletin C-461 is well illustrated with pictures, diagrams, cross-sectionals, and tables containing pertinent data.

26) BOTTOM-DUMP TRUCKS: Designed and built for off-the-highway hauling, *Euclid Road Machinery* bottom-dumps are described in a catalog which illustrates the improvements made. Closely hinged inside hopper doors when loading heavy excavation and cast steel hinges are just two of the advantages of these trucks.

27) CYANIDATION: Mineral Dressing Notes, No. 17 offered by *American Cyanamid Co.* is entitled "Chemistry of Cyanidation." A 40-page booklet, it is divided into four sections: Fundamentals of cyanidation, zinc in cyanidation, copper in cyanidation, and iron in cyanidation.

28) MINE CARS: The many types of cars, trucks, chilled tread wheels and repair parts are all described with illustrations in this booklet available from *American Car & Foundry Co.* A diagrammatic sketch illustrates the spring bumper arrangement.

29) FANS: A new engineering data book giving dimensions, performance, and prices on all types and sizes of fans is now obtainable from *Chelsea Fan & Blower Co., Inc.* Industrial pressure fans, duct booster fans, and automatic counter balanced shutters are just a few of the items discussed.

30) TRAMDRILLS: Single and double arm types are illustrated in dimensional drawings in bulletin SP-3004 distributed by *Chicago Pneumatic Tool Co.* The arm unit is mounted on roller bearings, making it possible to swing the arm 120° horizontally at floor level. Many other advantages are shown.

The Institute offers the second, completely revised, edition of this famed volume, for twelve years the leader in its field . . .

Industrial Minerals and Rocks

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One of the "100 Essential Titles" among Technical Books published in 1949-50, says R. R. Hawkins, chief of the Science and Technology Division, N. Y. Public Library, in the "Library Journal" for May 15, 1950.

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Industrial Minerals and Rocks is a "must" book for the modern engineer. Publication of this second, completely revised, 1156-page edition accents our present-day realization of the fact that young engineers with specific training in the field are now much in demand.

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The Use of Spiral Classifiers as Ball Mill Feeders

by T. C. King

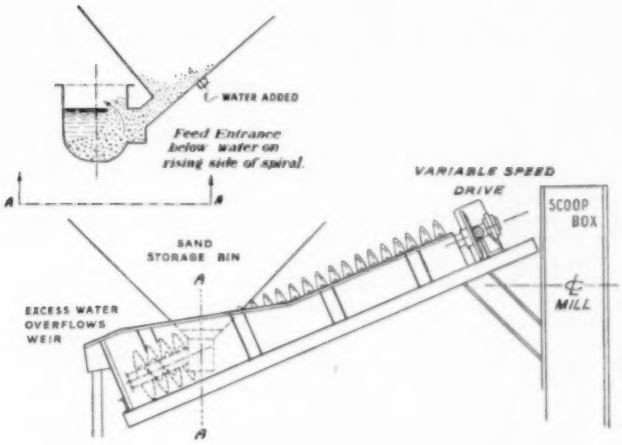

Fig. 1—Diagram of 24-in. spiral classifier used to feed wet sand from storage.

AT the new Graham-Central Mill of Eagle-Picher, near Galena, Ill., material is simultaneously dewatered and introduced into the ball-mill scoop boxes by the use of variable-speed, 24-in. spiral classifiers. This arrangement has proved satisfactory in that the moisture content of the wet feed is reduced, thus allowing maintenance of density within the mills, and a constant feed at any desired rate to the mills is provided.

In this mill the primary feed is wet crushed to minus ½ in. and preconcentrated by jigs to produce a bulk lead-zinc-iron concentrate and a waste tailing. The jig concentrate is then dewatered by a conventional dewatering classifier and stored in bins ahead of the ball mills. Even though this material is well dewatered, the discharge at the bin bottom is sloppy because of the gradual drainage of the water from the surface of the particles. Conventional belt feeders with vertical cutoff gates are unsatisfactory because of the flushing characteristic of the wet sand.

The new installation employs 24 in. x 10 ft, 9-in. Simplex double-pitch spiral classifiers for feeders from the bin bottoms to the ball mill scoop boxes (fig. 1). These units are arranged with straight tanks and are driven by 5 hp motors through a U. S. Vari-Drive to provide a 4 to 1 speed ratio.

Each bin discharges through a chute into the rising side of the spiral. The flushing characteristic of the material from the bottom of the bin is utilized to maintain a steady feed. A small amount of water tapped into the bottom of the bin insures uniform feed to the classifier. Excess clear water drains over the weirs.

Feed rate to the mills is controlled by the variable-speed drives. The feed delivered by the spiral at a given speed setting is relatively dry, uniform, and steady, and as such is easily controlled by the operator. The classifiers are set on a pitch of 5 in. per ft and have a sand-raking capacity of from 8 to 32 tons per hr.

It is believed that the solution of this problem offers possibilities for the storage of deslimed rod-mill sands and other similar products of the industry.

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AIME Columbus Meeting, September 1949.

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The Action of Sulphide Ion and of Metal Salts on the Dissolution of Gold in Cyanide Solutions

by C. G. Fink and G. L. Putnam

The dissolution of gold by cyanide solutions was studied by determining the time required for the solvents to dissolve gold leaf. Minute traces, even 0.5 ppm, of sulphide ion retard the dissolution of gold, and this behavior cannot be accounted for by the presently accepted hypotheses involving oxygen-depletion or thiocyanate formation. On the other hand, traces of the salts of lead, bismuth, mercury, and thallium considerably accelerate the dissolution. The beneficial effect of lead is dependent upon the pH and cyanide ion concentration of the solution.

ALTHOUGH cyanide solvents are used very extensively for the recovery of gold from ores and concentrates, the normal rate of solution of the precious metal as determined by Barsky, Swainson, and Hedley,[1] and others[2] is less than about 3 mg per sq cm of exposed gold surface per hour, corresponding to a corrosion depth of approximately 1.5 microns (0.00006 in.) per hr. Although the slow rate may be increased to some extent in certain cases, as, for example, when the the gold is partially imbedded in iron pyrites,[3] the leaching operation is generally the most time-consuming step of the cyanide process.

Frequently it is necessary for the leaching solvents to remain in contact with the ores for 50 to 200 hr in order to dissolve all of the gold particles.[4] Not only does the slow solution rate cause some increase in the capital costs, but the cyanide consumption also may be increased because of vaporization of hydrocyanic acid and secondary reactions with minerals such as malachite ($CuCO_3 \cdot Cu(OH)_2$), and pyrrhotite, which often are associated with gold. It is not surprising that numerous attempts have been made to accelerate the dissolution of gold in cyanide solutions.

Experimental

Gold Leaf Test: The gold leaf test was apparently originated by M. Faraday, who, by means of it, discovered that dilute cyanide solutions would dissolve gold readily in the presence of oxygen. The usefulness and reliability of the test have been discussed elsewhere.[5, 6] Essentially, the method consists of determining the time required for 5-ml portions of cyanide solvents to dissolve squares of 23 carat gold leaf, 0.5 cm on edge, when shaken in 10-ml test tubes with the solvents. The weight of gold leaf per test was 0.051 mg as determined by the direct weighing of a leaf 8.6x8.6 cm (weight = 15 mg) and measuring the area of the test sample (0.25 sq cm). Gold leaf is of remarkably uniform thickness.[5] Since gold leaf, like native gold, contains small amounts of copper and silver, the results are comparable with those one might expect in cyaniding gold ores. As six or more tests can be made simultaneously, the relative efficiencies of the solvents can be evaluated readily.

All dilute solutions of lead, bismuth, thallium, and mercury salts were prepared by the ordinary dilution method used by chemists. Our C.P. sodium cyanide and buffering reagents had no detectable amounts of either sulphide ion or heavy metals.

Soluble Sulphides in the Cyanide Process: The action of soluble sulphides is of interest in the study of the dissolution of gold in cyanide solutions. That gold ores which contain compounds of arsenic and antimony often are not amenable to direct cyanide treatment is well known, and it said that arsenic and antimony are "cyanicides."[7] It is commonly believed that part of the sulphur content of such ores is soluble in alkaline solutions to give sulphide ion, which reacts with the free cyanide content of the leaching solutions to form inert thiocyanates or reacts with the dissolved oxygen to form sulphites or sulphates.[8] A purpose of this paper is to demonstrate that sulphide ion may behave in a manner that cannot be explained by either the oxygen-depletion or thiocyanate formation theories and to prove directly that concentrations of sulphide ion which cannot be detected even by colorimetric methods may retard the dissolution of gold.

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A part of this paper is part of a dissertation submitted in partial fulfillment of the requirements for the degree of Doctor of Philosophy, Columbia University.

This paper is one of a series of publications dealing with the applied chemistry of gold (cf. ref. 5, 6, 7, and U. S. Patents 2,283,196 and 2,283,198). Other papers are in preparation.

AIME New York Meeting, February 1950.

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Courtney and Maguire[9] have studied the effect of sodium sulphide on the extraction of gold from its ores by solvents containing 3.1 g of sodium cyanide per liter. They gave no hypothesis to account for their observations, and the concentrations of sulphide ion in their solvents all exceeded 600 mg per liter.

Solvents were prepared which contained, per liter, 1.0 g of sodium cyanide, 10.6 g of sodium carbonate, 8.40 g of sodium bicarbonate, and variable amounts of sodium sulphide. The carbonate and bicarbonate are inert[5, 9] and were added to maintain a constant pH. When the efficiencies of the solvents were compared by the gold leaf method, the results shown in fig. 1 were obtained. It is noted that less than 0.50 mg of sulphide ion per liter is required to retard appreciably the dissolution of gold leaf. To oxidize this concentration of sulphide ion to sulphate, 1.00 mg of oxygen per liter would be required; yet cyanide solutions saturated with air at 25°C contain about 8 mg of oxygen per liter, and the oxygen solubility is not decreased by small percentages of dissolved salts.[5] Moreover, the oxygen contents of the solutions would have been brought rapidly to saturation by the agitation of the test tube. With regard to thiocyanate formation, this factor could have accounted for the loss of only 1.6 mg of sodium cyanide per liter from 1000 mg, and we have confirmed the fact that sodium thiocyanate[9, 10] does not retard the dissolution of gold.

It must be clear, therefore, that neither the oxygen-depletion nor the thiocyanate-formation theories can explain the behavior of sulphide ion. An important factor involved may be adsorption of sulphide ion by the gold at the metal-solution interface, thereby protecting the metal from corrosion by the solutions. Sulphides are harmful at concentrations below the ranges of the usual colorimetric methods.

Metal Salts in the Cyanide Process: The literature on the subject may be summarized as follows:

Lead Salts: According to Haden,[10] the use of lead compounds in large-scale operations at the Morro (Brazil) gold mine has permitted increased gold recoveries because of the buffering action of lead hydroxide and control thereby of the pH. But Haden overlooked the valuable accelerating effect of lead compounds on the dissolution of gold. Beyers[5] found that lead compounds render gold passive when the potassium cyanide concentration is below 0.08 pct. And Leaver, Woolf, and Jackson[11] have reported that lead salts are very deleterious in lime-sodium cyanide solvents. The present study soon disclosed that these earlier investigators apparently overlooked the true function of the lead salts and the best operating conditions.

Table I. Effect of Lead Nitrate on the Dissolution Velocity of Gold Leaf in Saturated Calcium Hydroxide Solutions of Sodium Cyanide

Mg of lead nitrate per liter of solution	500	100	10	0
Seconds time required to dissolve gold leaf	1,800	1,260	800	800

Table II. Effect of Cyanide Concentration on the Dissolution of Gold Leaf in Sodium Carbonate-Lead Nitrate Solutions

Composition of Solution	Seconds Time Required to Dissolve Gold Leaf
0.2 g NaCN per liter	220
0.2 g NaCN + 10 mg of $Pb(NO_3)_2$ per liter	130
0.5 g NaCN per liter	210
0.5 g NaCN + 10 mg of $Pb(NO_3)_2$ per liter	100
1.0 g NaCN per liter	200
1.0 g NaCN + 10 mg of $Pb(NO_3)_2$ per liter	50

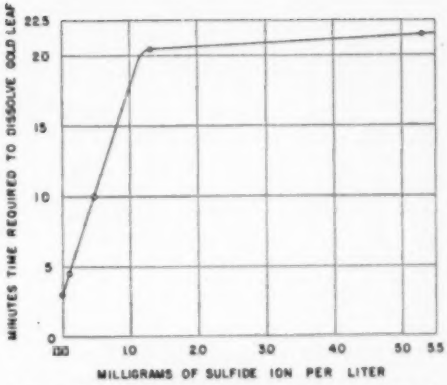

Fig. 1—Effect of sulphide ion on time required to dissolve gold leaf in a solution containing 1 g of sodium cyanide per liter.

Mercury Salts: The use of 5 to 10 mg per liter of mercuric salts as "oxidizing agents" in the cyanide process was patented in 1934 by Dowsett and John.[12] Keith (cited by Beyers[5]) found that mercury salts to the extent of 50 mg of mercury per liter of solution increased the rate of dissolution of gold. Keith (1890) believed that the mercury acted as an oxygen carrier by being redissolved continuously by the cyanide solution and precipitated by the gold.

Bismuth and Thallium Salts: No one has shown that these salts will accelerate the dissolution of gold under any conditions.

Recorded below are observations on the behavior of various metal salts on the time required to dissolve gold leaf. The results indicate that there are several metal salts, in concentrations of 1 to 40 mg per liter, which decrease the time required for cyanide solutions to dissolve gold leaf down to 0.5 to 0.032 of the time required in the absence of these metal salts.

Effect of Lead Nitrate Concentration: A solution was prepared which contained, per liter, 1.0 g of sodium cyanide, 1.06 g of sodium carbonate, and 0.84 g of sodium bicarbonate. The pH of this strongly buffered* solution is 9.9.[13] The results are plotted in fig. 2.

Referring to fig. 2, 0.1 mg of lead nitrate was used per liter of cyanide solution, and the time required to dissolve the gold leaf was reduced more than 30 pct. In the 5 ml of cyanide solution used in the test, there was 0.0005 mg of lead nitrate. Accordingly, 0.0005 mg of lead nitrate will accelerate the dissolution of 0.051 mg, or over 100 times its

* The Elsner equation[14] for the dissolution of gold in sodium cyanide is: $4Au + 2H_2O + O_2 + 8\,KCN = 4KAu(CN)_2 + 4KOH$. Consequently, the pH of an *unbuffered* cyanide solution dissolving gold does not remain constant. The sodium carbonates have no appreciable effect on the rate of dissolution of gold.[10]

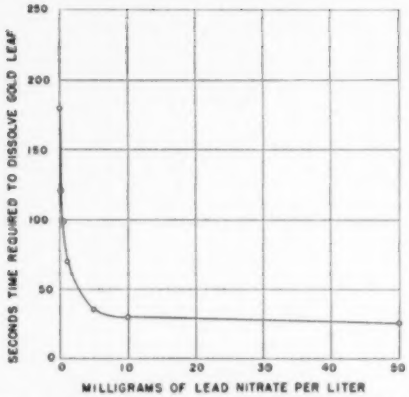

Fig. 2—Effect of lead nitrate on time required to dissolve gold leaf in sodium carbonate solutions containing 1 g of sodium cyanide per liter.

weight of gold. The observed acceleration cannot be attributed to the oxidizing power of the lead ion.

Effects of Calcium Hydroxide and of Sodium Carbonate on the Accelerating Effect of Lead Nitrate: A solution of 1.0 g of sodium cyanide in one liter of saturated limewater was prepared. The pH of this solution was about 12.7.[13] The results obtained when lead nitrate was added to this saturated solution of calcium hydroxide are shown in table I.

It is evident that in limewater solutions the addition of lead nitrate does not accelerate the dissolution of gold leaf. Quite the contrary, the lead compound increases the inhibiting effect[1] of calcium hydroxide. This is in accord with the reports of Leaver, Woolf, and Jackson[11] concerning the deleterious effect of soluble lead compounds in lime solutions.

The effect of increasing the sodium cyanide concentration in a limewater solution containing 10 mg of lead nitrate per liter was tested next. Referring to fig. 3, it is evident that the higher the cyanide concentration, the faster the gold leaf dissolves.

Effect of Sodium Cyanide Concentration on Dissolution of Gold Leaf in Sodium Carbonate Solutions: Tests were made next with a solution of somewhat lower pH (table II). All solutions were 0.02 mol (2.12 g per liter) with respect to sodium carbonate, and to alternate tests 10 mg of lead nitrate were added per liter of solution. The pH of 0.02 mol sodium carbonate is approximately 11.5.[12]

Table II shows that when lead nitrate is absent, change in the sodium cyanide concentration has little effect on the dissolution of gold leaf, and this is in accord with experiments of MacLaurin.[15] In the presence of lead nitrate, however, the sodium cyanide concentration has a very marked effect on the time required to dissolve gold leaf.

Effect of the pH of the Cyanide Solution: Although the accelerating effect of lead compounds had been definitely established in the case of sodium carbonate-sodium cyanide solutions, the results with the limewater cyanide solutions (fig. 3) required further study to account for the absence of the accelerating effect due to lead. It may be calculated that the solutions had a pH ranging between 12.3 and 12.6[16] and it was suspected that the high pH was to blame. Tests with sodium hydroxide in place of limewater supported this hypothesis—lead nitrate had no accelerating effect in sodium hydroxide-sodium cyanide solutions. An upper limit in pH of about 12 was established beyond which there was no acceleration in the dissolution of gold leaf in solutions containing 1.0 g of sodium cyanide per liter. The more dilute (less than 2 g per liter) solutions of sodium cyanide are the only solutions which are of interest from an industrial viewpoint.

With this upper limit of a pH of 12 in mind, a series of experiments with strongly buffered cyanide solutions of lower pH were made, the values being estimated from tables compiled by Kolthoff and Rosenblum.[17] That the buffering agents used have no affect on the dissolution of gold has been shown by other workers[1, 2, 9] and has been confirmed by applying the gold leaf solution method. The use of buffering agents was chosen in preference to other methods for three reasons:

1. Measurements with the antimony electrode are vitiated by the presence of traces of metals which the antimony is capable of displacing. As little as 0.1 mg of copper per liter, for example, may be a source of great error.[18]

2. In the absence of buffering agents, the pH of a cyanide solution dissolving gold rises as the gold dissolves, in accordance with the Elsner equation:

$$4\,Au + 8\,KCN + O_2 + 2\,H_2O = 4\,KAu(CN)_2 + 4\,KOH.$$

3. Because of the high pH of our solutions and the presence of sodium salts, the glass electrode method was not believed applicable for our purpose.

Table III indicates that lead is effective over the pH range of 8.0 to more than 9.9. Without lead nitrate additions, as recorded in table II, the dissolution time was about 200 sec.

Table III indicates that the effective pH range for mercury is 7.6 to more than 14. In other tests, not shown, it was found that the effective concentration of mercury, as mercuric chloride, is from less than 0.1 mg to more than 50 mg per liter.

Salts of bismuth and thallium were likewise found to have a marked accelerating effect. The results of table IV were obtained with a solution containing 1.0 g of sodium cyanide per liter.

Table IV indicates that bismuth and thallium compounds accelerate the dissolution of gold.

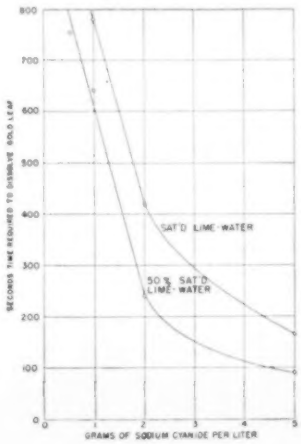

Fig. 3—Effect of sodium cyanide concentration on time required to dissolve gold leaf in lime water containing 10 mg of lead nitrate per liter.

Table III. Dissolution of Gold Leaf in Buffered Cyanide Containing Lead Nitrate and Mercuric Chloride

Concentrations in Grams per Liter, of Salts in Buffered Solutions Containing 1.0 G of Sodium Cyanide per Liter	Estimated pH	Seconds Time Required to Dissolve Gold Leaf in the Presence of	
		10 Mg of $Pb(NO_3)_2$ per Liter	10 Mg of $HgCl_2$ per Liter
40 g NaOH	14	200	80
10.6 g Na_2CO_3 + 8.4 g $NaHCO_3$	9.9	55	80
3.71 g H_3BO_3 + 2.97 g Na_2CO_3	9.0	85	60
4.95 g H_3BO_3 + 1.06 g Na_2CO_3	8.5	140	140
12.4 g H_3BO_3 + 1.06 g Na_2CO_3	8.0	130	140
21.4 g Na_2HPO_4 + 2.76 g KH_2PO_4	7.6	270	90

Many other metal salts, including those of zinc, cadmium, copper, tin, and magnesium, have been investigated. The salts of only four metals, lead, bismuth, mercury, and thallium, have been found effective.

Discussion of Results

Calculations of electrode potentials in cyanide solutions[5] show that gold metal can actually displace the ions of lead, bismuth, mercury, and thallium under all conditions that might obtain in cyanide practice. Of the common metals, only these four metals are more noble than gold in cyanide solutions. Lead ion is most readily displaced by gold in the pH range of 8 to 12; at higher and lower pH, the electrode potential of lead approaches that of gold in cyanide solutions.[6] At pH of less than 8, the cyanide ion concentration becomes very small; at high pH, the lead ion concentration decreases. Perhaps gold dissolves more rapidly because its surface character is altered by alloying with the displaced metals. Putnam[17] showed that zinc would alloy with displaced copper at room temperature, and the diffusion of gold into lead at room temperature is well known. The obvious voltaic couple explanation (i.e., the action of copper in promoting the solution of zinc in acid) is not believed applicable in this case. If diffusion of oxygen to the gold-solution interface, not the kinetics of the reaction, were the controlling factor, a small deposit of metal would not alter the rate of solution, as is proved by the film theory of chemical engineering.[18] The kinetics of the surface reaction probably control, and the displaced metals probably cause, acceleration of the rate-controlling reaction at the gold-solution interface.

Conclusions

1. Sulphide ion, even at concentrations below 0.5 mg per liter, may have a definite retarding effect on the dissolution of gold leaf, and this heretofore unknown deleterious action of minute traces (less than 1 mg per liter) of sulphide ion cannot be accounted for by either the oxygen-depletion or thiocyanate-formation theories. It is believed that the formation of an insoluble aurous sulphide film on the surface of the gold is responsible for the behavior of sulphide ion in protecting the gold from corrosion.

2. Salts of the four metals, lead, bismuth, mercury, and thallium, were found to decrease greatly the time required for sodium cyanide solutions to dissolve gold leaf. In some instances, less than 1 ppm (1 mg per liter) of metal salt had a definite effect.

Table IV. Acceleration of Dissolution of Gold with Salts of Bismuth and Thallium

Mg of Salt Added per Liter of Solution	Seconds Time Required to Dissolve Gold Leaf, Sec	
	Satd. $Ca(OH)_2$	0.2 Mol Na_2CO_3
No salts of bismuth or thallium	780	210
TlCl, 40 mg per liter	90	150
TlCl, 300 mg per liter	90	45
$TlCl_3$, 40 mg per liter	60	45
$TlCl_3$, 300 mg per liter	25	45
$Bi(NO_3)_3$, 30 mg per liter	50	90
$Bi(NO_3)_3$, 300 mg per liter	50	90

3. The beneficial effect is apparently limited to these four metals.

4. In the case of lead nitrate, increase in the sodium cyanide concentration above 0.2 g per liter increases the rate of dissolution of gold leaf in calcium hydroxide and in sodium carbonate solutions. The optimum pH range is about 8 to 11.5.

5. A lower limiting concentration of contact metal ion is necessary to bring about the accelerating effect. Thus, if the pH of the cyanide solution is above about 11.5, the lead ion concentration is too low to be effective. Similarly, the effect disappears if the cyanide ion concentration is too low.

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The Cleaning of Fine Sizes Of Bituminous Coals By Concentrating Tables

by R. E. Zimmerman

Wide attention is being placed upon various methods for cleaning the fine sizes of bituminous coals. The author describes and analyzes the results achieved on wet concentrating tables of modern design on unclassified feeds of —½ in. particle sizes, including tests on high-sulphur coals.

THE increased economic necessity of cleaning the small sizes of coal has forced wide attention upon various methods or processes being used. Although numerous cleaning plants in the past have been cleaning coal smaller than ½ or ⅜ in., the large quantities of higher ash fines being produced by mechanical mining and the need for cleaner coal under competitive market conditions have brought about many new installations of fine-coal cleaning plants.

Of various methods used, one of the more popular types is the wet concentrating table. Approximately 525 tables have been installed in the United States in the past four years of which 320 have been installed in 20, large new plants. There are now probably 1600 tables of all types in existence in this country for cleaning coal.

The purpose of this paper is to describe the cleaning of fine coal by wet concentrating tables. Its scope is confined to a discussion of the theory of tabling and some of the factors entering into tabling action, a description of the Deister SuperDuty diagonal-deck coal table and a discussion of the results of tabling various fine sizes of coals. Although tables have been used for cleaning coal as coarse as 2 in. and favorable results are reported, this paper is limited to reporting on the results of cleaning coal fines from ½ in. down. Also, although it is well known that the performance of tables is even superior under classified or closely sized feed conditions, this paper concerns itself with the results achieved by tabling unclassified feeds.

History of Wet Concentrating Tables

The wet concentration of minerals by some form of tabling is quite old, starting originally from fixed, smooth surfaced, tilted boards taking advantage of the laminar flow of liquid films and the separation of minerals according to specific gravity. This was followed by the development of rough deck surfaces

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induced by grooves or riffles and by the final achievement of adding a reciprocating action or differential motion to the table deck.

The development of tables occurred first in the separation of minerals other than coal, particularly in the concentration of heavy metallic minerals. There is an endless variety and, like other processes, it has required lengthy periods of evolution before achieving its final efficient form. Ore dressing men are familiar with the application of the early vanners, circular stationary Linkenbach tables, multiple round tables such as the Anaconda-Evans, and bumping or shaking tables such as the Rittinger or the Gilpin County concentrator.[1]

The advantage of some form of shaking motion was visualized early with the imparting of a reciprocating or shaking action by means of eccentrics or cams and by supporting the table deck on stilts or from hangers. The advantage of a differential action was found. This was first induced by a bumping action against a "bumping block" and later by a differential head motion applied through various designs by means of pitman and toggle actuated through an eccentric or crank shaft.

Some of the later designs of concentrating tables used in ore dressing are those of Garfield, Butchart, Deister-Overstrom, and Wilfley.[2] The first record of tables being used for coal in the United States appears to be of the bumping type called the Campbell Bumping Table. It was described by Phillips in 1893 when he wrote of the installation at Earlington, Ky.[3]

Campbell Bumping Tables have been used for many years at the Rosedale Washery, Johnstown, Pa., and their use has been described by Richardson.[4]

Practically all tables now used for coal cleaning use a driving mechanism, producing a differential motion, a rectangular or rhombohedral shaped deck, and some form of riffling on the deck surface. Possibly 95 pct of all tables being used for wet washing of coal in the United States are either the Deister SuperDuty diagonal-deck type or the Plat-O type. The former is the one most commonly used.[5] Until 1940 tables were more widely used on anthracite coals, but, with the current upswing of their use on bituminous coals, this condition has been reversed. For the past few years, ratio of placements have been approximately 3 to 1 in favor of bituminous coals.

Theory of Tabling

A great deal has been written concerning the theory of tabling and the various physical laws which enter into its operation.[2, 3-5] Much more work has yet to be done in studying thoroughly the numerous variables that can be made in tabling and the interrelation of the physical laws that affect separation.

Gaudin's[2] mathematical analysis of flowing film concentration is of fundamental value and a worthy contribution. His analysis pertains mostly to smooth deck surfaces and does not go into detail concerning the phenomena surrounding the use of riffles. Bird and Davis,[4] Yancey[5, 9] and Gandrud[6] have made valuable contributions to the study of the various factors affecting tabling of coal.

Separation of particles on a table deck is not merely the result of their differences in specific gravity, but rather there are numerous interrelated phenomena. A decided advantage of tabling over

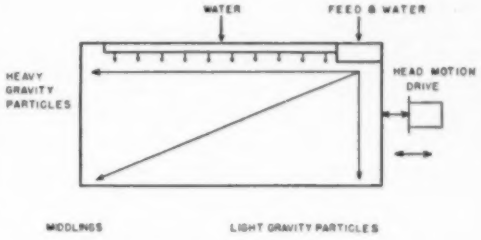

Fig. 1—Natural path of particles on a table.

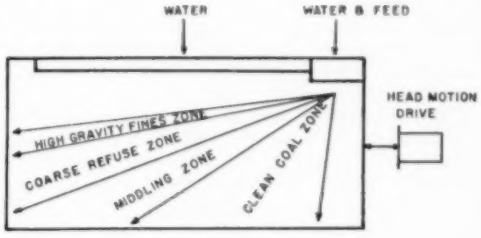

Fig. 2—Zones of separation on a table.

some other known processes is the fact that a great many variables can be made to aid and assist in effective separation.

Basically the separation of different minerals on a concentrating table are the result of the following factors: (1) Specific gravity of the particles, (2) size of particles, (3) shape of particles, (4) stratification (or consolidation trickling), and (5) hindered settling.

The effective use of these factors may be controlled and aided by the following table adjustments: (1) Differential action of the drive or head motion, (2) speed of motion, (3) length of stroke, (4) side tilt, (5) end elevation, (6) size of deck area, (7) shape of deck, (8) character of deck surface, (9) riffle design, (10) amount and distribution of

Fig. 3 — Deister SuperDuty diagonal deck No. 7 coal washing table.

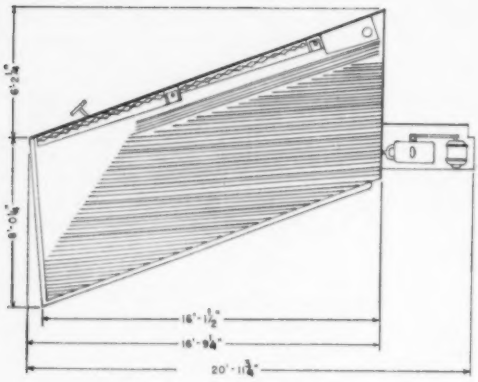

Fig. 4—Plan view of Deister table deck.

wash water, (11) feed water-solids ratio, (12) feed rate, and (13) feed uniformity.

Concentrating tables are essentially a rectangular or rhombohedral shaped deck supported by suitable frame and bearings, generally sloping in the direction of the short axis and usually tilted upwards at the refuse end of the long axis and differentially reciprocated in the direction of the long axis.

This action of head motion tends to move the particles longitudinally towards the end of the table, while at right angles to this movement wash water is distributed along the high side of the table which tends to move particles down the slope parallel to the short axis of the table. Hence, the natural path of a particle on a concentrating table would be from the feed-box corner to the diagonally opposite corner of the deck as shown in fig. 1.

There is a natural tendency for heavier particles to sink to the deck of the table and for lighter particles to stay nearer the surface. As a consequence, the heavier particles are carried farther along the longitudinal axis since, as they are on or near the surface of the deck, they are more affected by the reciprocating action of the table. The lighter particles, as they are at or near the top of the moving bed or particles, travel slower in a longitudinal direction and thus become separated from the heavy particles.

Wash water and the diluting water in the feed cause the movement of the lighter gravity particles down the short axis of the table. Thus the feed material fans out from the feed box, where it enters on to the table with heavy gravity particles moving in one direction and lighter particles moving in another. By proper adjustments of all the factors previously mentioned, a clear-cut separation is possible between high and low-gravity particles, or between refuse and coal.

Riffles are of great importance to tabling and are responsible for their considerably greater capacity over smooth decks and for greater accuracy of separation. Behind the riffles, particles are delayed in their downward movement along the slope of the deck. Hindered settling, consolidation trickling, and other phenomena occur as the particles tumble over the riffles and stratification ensues. The heavier particles are settled to the deck of the table and trapped behind the riffles thus insuring their positive movement in the direction of the end of the table.

Since stratification and separation of particles are not complete as a result of any one riffle, a series of riffles are used, repeating the cycle of stratification and hindered settling from riffle to riffle, obtaining purer products as the particles fan out and progress forward and downward over the table.

The smallest heaviest particles stratify themselves downward to the surface of the deck more quickly and are carried out by table movement toward the refuse end at a faster rate than coarse heavy particles. Light-gravity larger pieces ride on the top layer of particles and flow on down the slope of the deck as a result of the cross flow of wash water at right angles to the shaking movement of the table. Heavy large particles settle down next to the fine heavy particles and are carried behind the riffles towards the longitudinal end but do not ride as high up the table as the heavy fine particles.

There is a tendency for the fine or small size light-gravity particles to lie between the coarse heavy particles and work over into the zone with the coarse refuse, and, if the only separating action on a table were that of straight stratification, this would result in fine coal getting over into the refuse. Fortunately, however, hydraulic classification occurs under hindered settling conditions which tends to counteract this and by proper use of water dilution and wash water as well as riffle design the small light particles of coal are washed off with coarse light particles or coarse coal.[6, 10]

When there is a definite clear-cut gravity difference between coal and refuse, a wide size range or an unclassified feed may be treated very efficiently. When there is a large proportion of material intermediate in gravity between coal and refuse, the separation of an unsized feed will become increas-

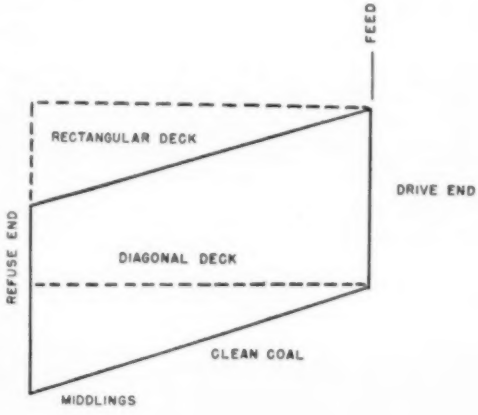

Fig. 5—Comparison of diagonal deck with rectangular deck.

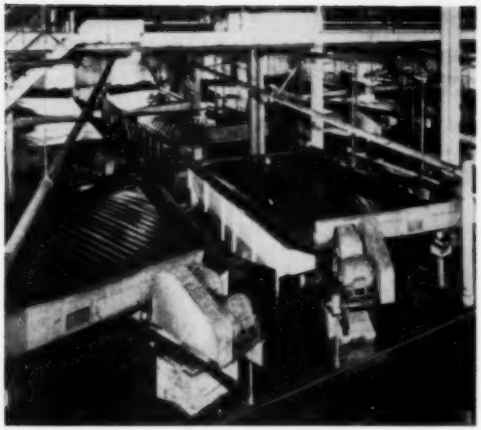

Fig. 6—Installation of Deister No. 7 coal washing tables in a modern coal washing plant.

ingly difficult. Fig. 2 illustrates the zone where this material will fall.

In only rare instances, on coals of very difficult cleaning characteristics, is it necessary to cut a middling or a third product from a table. When it is done, these middlings are normally disposed of as a secondary fuel since little can be accomplished by retabling such material.

The advantage of a classified feed where a large proportion of middlings or near gravity material exists is that the danger of having small light-gravity particles of coal get over into the refuse or middling zone vanishes. Sharper lines of demarcation can be made, which result in very high efficiencies. B. M. Bird and H. F. Yancey[8, 9, 11, 12] have demonstrated many times the advantages of classified feed to the tables where coal is difficult to treat because of high percentages of near gravity material or where middlings from another cleaning process are to be cleaned on tables.

For most coals, however, it has been demonstrated that under proper conditions of operation an unclassified feed may be used for cleaning sizes ranging from ½ in. to 0, and produce products containing relatively few misplaced particles. On the extreme fines, separation may be effective on particles as small as 200-mesh with reasonably efficient recoveries on the 100x200-mesh sizes.

Brief mention should be made of the effect of particle shape in tabling. Generally speaking, flat particles have a tendency to pass over with the refuse. Flat particles adhere to the table and have a tendency to move along with heavier gravity particles. Fortunately, in most instances coal particles are cubical or splintery and many refuse particles are naturally flat in shape. H. F. Yancey,[7] who conducted an investigation on the effect of particles shape, states, however, that "shape of particle is a factor of minor importance in tabling of most unsized coals insofar as the overall efficiency of the process is concerned."

An unmistakable advantage of a concentrating table which should be mentioned is that the products being made are always visible. The operator can observe his refuse, clean coal, or middlings and can note quickly the effect of any change in feed conditions or table adjustments.

Description of the Deister Diagonal-deck Table

Although there are other wet concentrating tables in use, the writer's experience and test performance data concerns the Deister SuperDuty diagonal-deck table and a brief description of this particular table is in order.

The current Deister table was developed from the widely used Deister-Overstrom diagonal-deck table. Improvements in design included greater ruggedness and Concenco Anti Friction Bearing Headmotion. This motion is similar in principle to the Overstrom, being of the pitman and toggle design, but it is of stronger construction with ball and roller bearings lubricated by a circulating oil system. The table most often used for coal cleaning is their No. 7, SuperDuty size, either linoleum or rubber covered and with wooden or rubber riffles. The bulk of the recent tables are rubber covered with rubber riffles.

Fig. 3 shows a photograph of the No. 7, coal washing table. It can be made either right hand or left hand to suit conditions in the plant. The head motion can be operated from a line shaft with belt and pulley, but invariably new installations use the individual drives. The No. 7 table utilizes a 3-hp motor, although power consumption is seldom over 2 hp.

Fig. 4 shows the general dimensions and plan view of the coal washing table with one type of riffle pattern. Other riffle designs can be used. The most common are the line of motion type in which all the riffles are in a diagonal line parallel to the motion of the table.

Riffling is characterized by a large number of shallow riffles, interspersed with higher strips, which provide pools to settle finer sized particles. All riffles are high at the feed end and taper or feather

Fig. 7—Closeup view of Deister table showing launder arrangements for handling products.

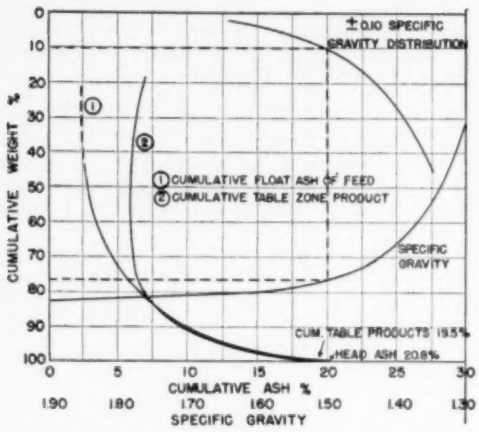

Fig. 8—Case study "A" washability of ½ in. x 0 table feed and cumulative table products.

towards the refuse end. The actual height of the riffles varies, depending on the sizes of coal being cleaned. The highest may reach 2 in. at the deep end.

Feed mixed with water is fed into the feed box in the upper right hand corner of the table illustrated in fig. 4. In addition to the water with the feed, water is also added at two points along the top edge of the table, where it runs through troughs equipped with adjustable louvers, permitting water to be distributed uniformly or in variable quantities along the length of the deck.

The table base is a pair of heavy parallel channels running diagonally to the opposite corner from the drive with the head motion and rocker arm mounted at one end on an extension of the channels. Another rocker and a helical compression spring are mounted at the other end of the channels. A subframe constructed of lighter channels and beams supports the table deck. This subframe straddles the upper edges of the main channels resting on saddles at four points along its diagonal line, and the two outer corners rest on blocks of which the elevations are adjustable by oppositely facing wedges manipulated by hand wheel and spur gearings. Maximum tilt on the No. 7 table is approximately 8°. The tilting mechanism has an operating range of 3°, which may be employed in any portion of the overall range. End or longitudinal tilt is obtained by shimming up the main channel supports at the spring or refuse end.

The table deck is built of cypress in strips laid diagonally to its framework. The deck is covered with either linoleum or rubber and riffles made of pine or hardwood tacked on, or if rubber riffles are used, glued on by a special rubber cement.

The speed of the Concenco head motion may be varied from 250 to 300 rpm depending upon the type of coal being washed. Stroke length is also adjustable, generally between the range of 7/16 to 1¼ in.

End tilt or end elevation of the table may be anywhere from 0 to 12 in. For ¼ to 0 coal one seldom operates higher than 3 in. The coarser the coal, the greater the end elevation. The side tilt, or slope, is adjusted by turning the hand wheel that operates the subframe tilting mechanism. One seldom operates at a greater slope than 6°.

The amount of water required will vary with particle size and other characteristics of the coal and refuse. In most cases, feed is at approximately 40 pct solids, and dressing water is added to bring the overall water to solids ratio to 2 to 1, by weight.

As the name implies, the deck dimensions are diagonal or rhombohedral in shape. This has advantages over the ordinary, rectangular shape. Fig. 5 illustrates this point. Generally speaking, the area of a table actually occupied by the coal and refuse takes on the form of a diagonal band from the feed to the refuse or high-gravity end. This is illustrated in fig. 1, where the lines of force are shown, and in fig. 2, where the zones of various products are shown.

Since the material tends to distribute in a diagonal direction, a table of the diagonal shape illustrated in fig. 5 more effectively uses deck area and can utilize more riffles, a desirable characteristic for higher tonnages, unclassified feed, and where there is a great deal of near gravity or middling material to be separated.

Fig. 6 is a photograph of one of the more recent installations of Deister coal washing tables. At this plant they are cleaning ¼ in. x 0 bituminous coal. Fig. 7 is a close-up photograph of one of the tables and shows launder arrangement for discharging products.

The capacity of the No. 7 coal washing diagonal-deck table will vary with the type of feed, being greater for coarser particles (up to the practical size that can be treated) than for extreme fines, greater for classified feeds than for unclassified, and, as in all processes, greater for coals containing a minimum of near gravity material.

Table I. Screen Sizes, Ash* and Sulphur* of ½ in. x 0 Feed to Deister Table

Size, Mesh	Direct Analysis, Pct			Cumulative Analysis, Pct					
	Wt	Ash	Sul.	Wt	Ash	Sul.	Wt	Ash	Sul.
+ ⅜ in. rd.	19.2	26.3	1.39	19.2	26.3	1.39	100.0	20.8	1.50
⅜x4	21.5	20.7	1.43	40.7	23.3	1.41	80.8	19.5	1.53
4x8	21.9	20.7	1.41	62.6	22.4	1.41	59.3	19.1	1.57
8x14	15.2	18.8	1.51	77.8	21.7	1.43	37.4	18.2	1.66
14x28	8.6	17.2	1.54	86.4	21.3	1.44	22.2	17.8	1.76
28x48	6.0	17.2	1.70	92.4	21.0	1.46	13.6	18.2	1.91
48x100	3.2	16.9	1.98	95.6	20.9	1.48	7.6	19.0	2.07
—100	4.4	20.6	2.13	100.0	20.8	1.50	4.4	20.6	2.13
Total	100.0	20.8	1.50						

* Analyses on "Dry Basis".

Table II. Case Study "A" Zone Analysis of Table Products from ½ in. x 0 Feed

Zone	Direct, Pct			Cumulative Recovery, Pct			Cumulative Reject, Pct		
	Wt	Ash	Sul.	Wt	Ash	Sul.	Wt	Ash	Sul.
1	23.4	6.6	1.05	23.4	6.6	1.05	100.0	19.5	1.50
2	16.5	5.2	1.14	39.9	6.0	1.09	76.6	23.5	1.64
3	23.9	5.8	1.29	63.8	5.9	1.16	60.1	28.5	1.78
4	15.4	8.8	1.61	79.2	6.5	1.25	36.2	43.5	2.10
5	2.0	16.4	2.43	81.2	6.7	1.28	20.8	69.2	2.47
7	1.9	41.6	2.95	83.1	7.5	1.33	18.8	74.8	2.48
8	16.7	78.5	2.36	99.8	19.4	1.49	16.9	78.5	2.42
9	0.2	79.6	7.59	100.0	19.5	1.50	0.2	79.6	7.59
Total	100.0	19.5	1.50						

On average Western Pennsylvania bituminous coals, capacities of 9 to 10 tph per table for ¼ in. x 0 size feed can very well be attained with excellent efficiencies. By increasing the maximum size to ⅜ in., the feed rate can be raised to 12 tph. With a classified feed, that is where there is a close size range of particles, the table capacity may be sharply increased to as much as 15 to 25 tph or higher. B. M. Bird and H. F. Yancey state that "classification (of table feed) approximately doubles the capacity of the table while permitting efficiencies equal to those of unsized feeds."[11]

Results of Test Data on Unclassified Table Feeds

To study the results of cleaning the fine sizes of bituminous coal, particularly under unclassified feed conditions, numerous tests have been made under a wide variety of conditions on many different coals. To illustrate the performance of the Deister Super-Duty diagonal-deck coal washing table, test data on four different coals are presented for analysis.

Since interest is also general concerning the effect of tabling coal of different size consists, the four case studies presented are tests of a wide variation of sizes. Examination of the washability curves and ash and sulphur analyses also show them to be considerably different from the viewpoint of their natural characteristics. For the sake of brevity, the numerous operating variables, such as table speed, stroke, side tilt, end elevation, solids-water ratio, etc. are not included in this paper. In general, it can be assumed that table settings were under optimum conditions.

Case Study "A"

The data shown under this study come from the testing of ½ in. x 0 Pittsburgh seam coal from Washington County, Pa. The table feed was obtained as a result of screening the run-of-mine coal on ½-in. round hole screens.

Zone samples were taken of the products from the table by marking off linear distances along the table, dividing it into 10 zones, with zones 1 through 5 taken along the coal side and zones 6 through 10 along the refuse end. In this particular test no particles came over in zones 9 and 10, which are along the upper side of the refuse end.

In sampling the zonal products, samples were taken simultaneously. A special zone sample cutter was arranged so that all the products from each zone were collected, each container being large enough to hold all the material passing into it. Each zone's product was then filtered, dried, and weighed. Table capacity calculated as 10.5 tph.

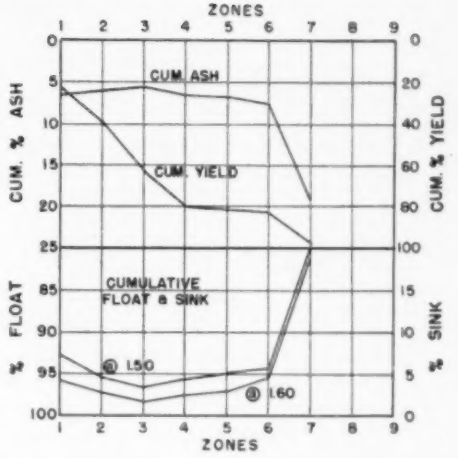

Fig. 9—Case study "A" ½ in. x 0 feed showing cumulative ash, yield, float-and-sink byproduct zones.

Table I shows the size distribution of the feed together with the ash and sulphur analyses of each size increment. Analyses are on a "dry" basis; sieve sizes are Tyler standard. One notes that this coal runs 20.8 pct ash and 1.50 pct total sulphur.

It is a relatively coarse feed containing 19.2 pct of ½x⅜ in. and with 77.8 pct as +14 mesh. The +⅜ in. is the highest in ash, being 26.3 pct and the lowest in sulphur content, 1.39 pct. The middle sizes are the lowest in ash percent.

Fig. 8, shows the washability curve of this feed, its ±0.10 sp gr distribution, as well as a curve (2) indicating the cumulative ash yield of the zonal products from the table. One could term this coal a moderately difficult coal to clean. The specific gravity distribution curve indicates that any washing below 1.50 sp gr would be difficult, the amount of near gravity material below 1.50 being quite high.

Assuming the lower practical gravity of separation as being 1.50, such a separation would give a theoretical ash in the clean coal product of 5.5 pct with a yield of 76.5 pct. At this same yield point the clean coal product of the table analyzed 6.4 pct. Also one observes that at 1.60 sp gr the cumulative zonal

Table III. Float-and-Sink at 1.50 Sp Gr on Table Products (½ in. x 0), Case Study "A"

Zone	Product. Wt. Pct	Float, Pct			Sink, Pct			Cum. Wt Pct Sink in Product
		Wt	Ash	Sul.	Wt.	Ash	Sul.	
1	23.4	92.9	5.0	1.02	7.1	27.7	1.47	7.1
2	16.5	98.7	4.7	1.11	1.3	36.7	2.84	4.7
3	23.9	98.7	5.4	1.26	1.3	35.6	3.46	3.4
4	15.4	91.1	6.2	1.40	8.9	36.3	3.76	4.4
5	2.0	72.9	7.9	1.77	27.1	39.2	4.29	5.0
6	1.9	30.7	7.6	1.63	69.3	56.7	3.54	5.4
7	16.7	7.7	6.5	1.44	92.3	84.5	2.44	20.9
8	0.2							*
Total	100.0							

* Not enough sample in zone 8 for analysis at 1.50.

Table IV. Float-and-Sink at 1.60 Sp Gr on Table Products (½ in. x 0), Case Study "A"

Zone	Product. Wt. Pct	Float, Pct			Sink, Pct			Cum. Wt Pct Sink in Product
		Wt	Ash	Sul.	Wt.	Ash	Sul.	
1	23.4	96.0	5.5	1.02	4.0	34.1	1.76	4.0
2	16.5	99.2	4.6	1.11	0.8	44.9	3.56	2.7
3	23.9	99.5	5.6	1.27	0.5	48.3	4.45	1.8
4	15.4	95.0	7.1	1.49	5.0	42.6	3.91	2.4
5	2.0	79.8	9.5	1.96	20.2	43.6	4.39	2.6
6	1.9	36.9	10.9	1.89	63.1	59.6	3.52	4.2
7	16.7	8.3	8.1	1.62	91.7	84.9	2.43	19.0
8	0.2							*
Total	100.0							

* Not enough sample in zone 8 for analysis at 1.60 shows 97.5 pct sink at 1.90.

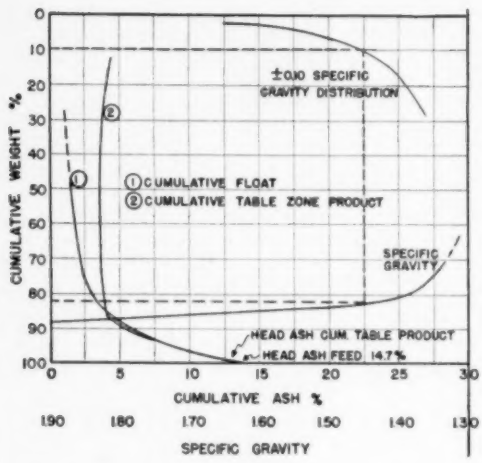

Fig. 10—Case study "B" washability of ¼ in. x 0 table feed.

curve approximates the theoretical curve almost perfectly.

Table II gives the analyses of the table products by zones. The zone 1 product is somewhat higher ash than the next two consecutive zones. This is characteristic, as this first zone near the feed end contains some high-ash fines carried over by the feed water.

The first five zones produced 81.2 pct of the product and ran 6.7 pct ash. The refuse, that is, the products from zones 6, 7 and 8, ran 74.8 pct ash.

One realizes, of course, that the entire product on the long side of the table, generally called the "coal" side, does not have to be sluiced to clean coal. The products from the zones near the refuse can be directed to refuse or separated out as a third product.

Although sulphur was not considered the major problem in this test, it is noted that the cumulative sulphur of zones 1 through 5 ran 1.28 pct, that from zones 1 through 3 ran 1.16 pct. The inherent sulphur of this particular coal is approximately 0.95 pct.

Tables III and IV show the results of float-and-sink analyses at 1.50 and 1.60 sp gr, respectively, of the various product zones. At 1.50 sp gr which, as has been pointed out previously, is the lower limit for practical washing, the first five product zones contain 5.0 pct sink. At 1.60 this is reduced to 2.8 pct.

Table V. Screen Size and Ash Analysis of ¼ in. x 0 Feed to Deister Table, Case Study "B"

Size, Mesh	Direct, Pct		Cumulative, Pct			
	Wt	Ash	Wt	Ash	Wt	Ash
+4	6.9	26.0	6.9	26.0	100.0	14.7
4x8	23.5	18.5	32.8	20.1	93.1	13.9
8x14	18.0	13.0	50.8	17.6	67.2	12.1
14x35	26.7	11.5	77.5	15.5	49.2	11.8
35x100	13.9	11.1	91.4	14.8	22.5	12.2
—100	8.6	13.9	100.0	14.7	8.6	13.9
Total	100.0	14.7				

Fig. 9 shows graphically the cumulative ash percent, yield percent and float-and-sink at 1.50 and 1.60 sp gr for the products from the various zones along the table discharge, beginning with zone 1 and cumulating the products up through zone 8.

Case Study "B"

The data shown under this study are the results of testing ¼ in. x 0 screened run-of-mine from the Pratt seam, taken from a mine near Birmingham, Ala.

Zone samples were taken along the table as in study "A", except that the long or coal side was divided into six zones and the refuse end into three with zone 7 at the diagonal corner. Feed rate of the table at the time samples were taken was 10.0 tph.

Table V gives the size distribution of the feed together with the ash analyses of the size increments. This coal is finer and lower in ash than the coal of the previous study. Only 6.9 pct of it is +4-mesh, while 8.6 pct is —100-mesh. The calculated head ash of this coal is 14.7 pct. The +4-mesh is by far the highest ash increment, running 26.0 pct. The —100-mesh size contains only 13.9 pct ash.

Fig. 10 shows the washability curve of this Pratt seam ¼ in. x 0 table feed, the ±0.10 sp gr distribution, as well as a curve (2) indicating the cumulative ash yield of the products from the zone samples mentioned above.

Table VI. Case Study "B" Zone Analysis of Table Products from ¼ in. x 0 Feed

Zone			Cumulative Recovery, Pct		Cumulative Reject, Pct	
	Wt, Pct	Ash, Pct	Wt	Ash	Wt	Ash
1	11.8	4.6	11.8	4.6	100.0	13.8
2	40.6	3.4	52.4	3.7	88.2	15.1
3	27.1	4.7	79.5	4.0	47.6	25.0
4	4.0	8.6	83.5	4.3	20.5	51.9
5	2.6	13.1	86.1	4.5	16.5	62.4
6	0.7	17.1	86.8	4.6	13.9	71.6
7	0.6	30.3	87.4	4.8	13.2	74.5
8	9.8	74.2	97.2	11.8	12.6	76.6
9	2.8	85.0	100.0	13.8	2.8	85.0
Total	100.0	13.8				

This coal is not only a lower ash coal but compared to case study "A" is a much easier coal to clean. The ±0.10 sp gr curve shows that the lower limit for efficient cleaning is at 1.45, where a theoretical ash in the float coal is only 3.5 pct producing an 82 pct recovery. At this same yield point the cumulative zone products from the Deister table gave 3.8 pct ash, a very close check with the theoretical. This gap, of course, widens beyond this point at the lower gravities.

Table VI gives the zone analysis of table products in case study "B". As is general, the zone 1 showing 4.6 pct ash gives a somewhat higher ash percent than the next zone, which produced a 3.4 pct ash product. Zones 1 through 6 gave a cumulative ash of 4.6 pct with a combined yield of 86.8. Even adding the corner zone 7 into the clean coal side brought the overall clean coal ash only to 4.8. Refuse or reject ash combining zones 7, 8, and 9 was 74.5 pct.

In table VII, float-and-sink analyses are shown for each zone product at 1.40 and 1.60 sp gr. The combined products of zones 1 through 6 show 13.7

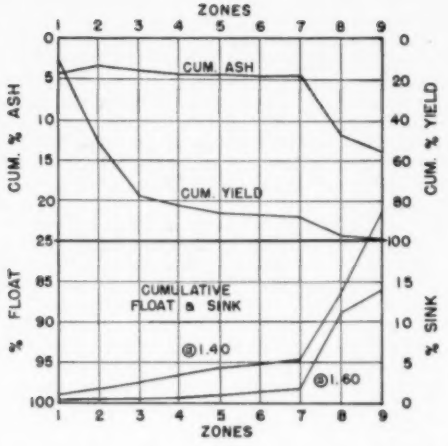

Fig. 11—Case study "B" ¼ in. x 0 feed showing cumulative ash, yield, float-and-sink byproduct zones.

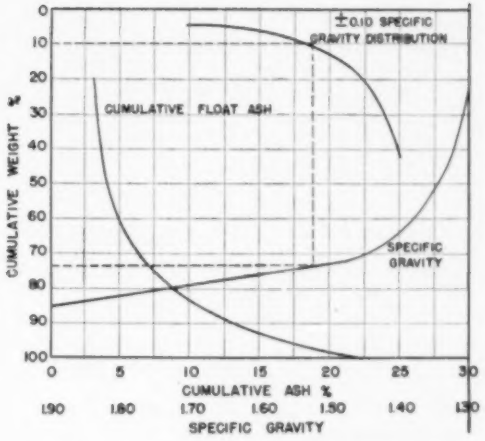

Fig. 12—Case study "C" washability 3/16 in. x 0 table feed.

pct sink at 1.40 and only 2.6 pct sink, however, from zones 1 through 4. Since 1.40 sp gr is lower than the desirable minimum gravity according to the ±0.10 sp gr distribution curve, it is logical that one would run into difficulties near the middling zone of the table.

Testing these same products at 1.60 sp gr, less than 1.0 pct sink is observed in the cumulated products of zones 1 through 5 and, including the sixth zone, only 1.05 pct sink was found. Combining the refuse zones 7-8-9, one finds these zones contain 5.3 pct float coal at 1.40 and 6.9 pct float coal at 1.60 sp gr.

Fig. 11 shows graphically the results plotted from the accompanying test data and gives the cumulative ash and yield percent as well as the cumulative float-and-sink percent at 1.40 and 1.60 sp gr, cumulating the products from zones 1 through 9.

Case Study "C"

This study is on 3/16 x 0 coal from the Ohio No. 8 seam in Jefferson County of eastern Ohio. The table feed is a mixture of 80 pct total seam deep mined and 20 pct strip coal and is the resultant product of screening the cleaning plant raw feed on 3/16 in. round hole screens.

Zone samples were not taken at various points along the table. Rather, the entire product from the long or coal side of the table was considered as the clean coal product and the material discharged off the end as refuse. Screen and ash analyses of the feed, clean coal, and refuse were taken and thus a comparison can be made of ash reduction at various size increments. Table feed rate at the time samples were taken was 8.7 tph.

Table VIII gives the size consist together with the ash analyses of the various sizes. This is a dirty coal, the calculated head ash being 22.2 pct. Unlike some of the other coals reported in this paper, the coarsest increment, namely the 3/16 in. x 8 mesh, had the lowest ash, being 14.8 pct. The —100-mesh contained 29.8 pct and the —200-mesh, 31.7 pct ash.

The table feed was considerably finer than those in studies "A" and "B", containing 10 pct —100-mesh and 5.2 pct —200-mesh particles. These extreme fines are difficult to clean by any gravity process.

Fig. 12 shows the washability curves for the table feed. The ±0.10 sp gr distribution curve indicates that a gravity of 1.52 must be reached before one finds less than 10 pct of the particles between a 0.10 difference in sp gr, an amount generally accepted as the maximum limit for good separation. Of the feed 12 pct lies between 1.40 and 1.60 sp gr. The theoretical yield at 1.52 is 74.0 pct, giving a cumulative float-ash percent of 7.0.

Table VII. Float-and-Sink at 1.40 and 1.60 Sp Gr on Table Products from ¼ in. x 0 Feed, Case Study "B"

Table Zone	Product Wt, Pct	At 1.40 Sp Gr[a]				Cum. Sink, Pct	At 1.60 Sp Gr[a]				Cum. Sink, Pct
		Float, Pct		Sink, Pct			Float, Pct		Sink, Pct		
		Wt	Ash	Wt	Ash		Wt	Ash	Wt	Ash	
1	11.8	99.0	2.4	1.0	19.4	1.0	99.9	2.5	0.1	44.1	0.10
2	40.6	98.3	2.7	1.7	17.6	1.6	99.8	2.9	0.2	35.2	0.18
3	27.1	95.1	3.1	4.9	20.2	1.7	99.6	3.8	0.4	35.7	0.25
4	4.0	79.6	3.5	20.4	34.9	2.6	96.2	6.7	3.8	49.5	0.42
5	2.6	69.5	3.6	30.5	31.8	11.7	84.8	7.3	15.2	39.5	0.86
6	0.7	61.9	3.7	38.1	35.0	13.7	76.3	7.4	23.7	42.3	1.05
7	0.6	32.0	4.4	68.0	40.9	18.3	48.1	11.1	51.9	46.2	1.40
8	9.8	4.0	4.4	96.0	77.3	26.1	6.4	12.6	93.6	77.9	11.05
9	2.8	0.0	0.0	100.0	85.1	28.9	0.2	16.9	99.8	85.2	13.54
Total	100.0										

[a] Float-and-sink analyses on ¼ in. x 100-mesh, data do not include the —100-mesh.

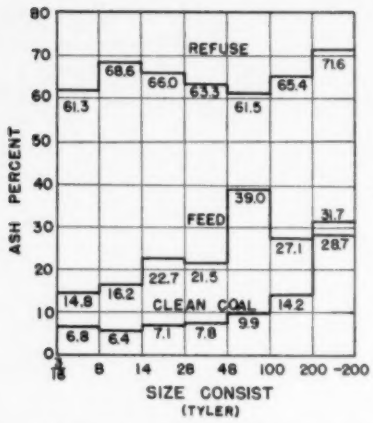

Fig. 13—Case study "C" 3/16 in. x 0 feed, showing ash distribution according to size consist.

Table IX shows the screen sizes of the table products with ash analyses. The calculated head ash of the clean coal was 9.4 pct with 67.3 pct for the refuse. Based on the formula that

$$\text{Recovery, } R = \frac{t - f}{t - c} \; 100$$

where t is the ash of the refuse, f is the ash of the feed, and c is the ash of the clean coal product, the recovery or yield was 78.0 pct. Examining the theoretical yield of 78.0 pct on the raw feed washability curve in fig. 12, one notes that the float coal shows a theoretical ash of 8.0 pct.

Again examining the results shown in table IX, one finds that the cleanest increment in the clean coal product is the 8x14-mesh which analyzed 6.4 pct ash. The highest ash is in the —200-mesh analyzing 28.7 pct. Cumulating the sizes down to 28-mesh, one obtains a 7.0 pct ash product. Cumulating the sizes down farther to 100-mesh, one obtains a 7.4 pct ash product, and at the 200-mesh level 7.7 pct ash results. Obviously the —200-mesh is the offender in bringing up the composite ash of the total clean coal to 9.4 pct.

It is considered noteworthy that the 48x100-mesh coal has been reduced from 29 pct ash to 9.9 and the 100x200-mesh increment from 27.1 pct ash to 14.2 pct.

Table X records the results of float-and-sink analyses of the clean coal and refuse at 1.50 and 1.60 sp gr only on the +100-mesh sizes because of the difficulty of float-and-sink testing of the —100-mesh sizes. As such, it is only partly indicative of the actual results. One notes that the clean coal +100-mesh had 3.5 pct sink and the refuse had 4.7 pct float material at 1.50 sp gr. At 1.60 sp gr there was 1.8 pct sink in the clean coal and 8.5 pct float in the refuse. Fig. 13 illustrates graphically the ash distribution, by various size increments, in the table feed, clean coal, and refuse.

Table VIII. Screen Sizes and Ash Analyses of 3/16 in. x 0 Feed to Deister Table, Case Study "C"

Size, Mesh	Direct, Pct		Cumulative, Pct			
	Wt	Ash	Wt	Ash	Wt	Ash
3/16x8	16.8	14.8	16.8	14.8	100.0	22.2
8x14	25.9	16.2	42.7	15.7	83.2	23.9
14x28	20.6	22.7	63.3	17.9	57.3	27.4
28x48	13.9	21.5	77.2	18.6	36.7	30.0
48x100	12.8	29.0	90.0	21.0	22.8	35.0
100x200	4.6	27.1	94.8	21.2	10.0	29.8
—200	5.2	31.7	100.0	22.2	5.2	31.7
Total	100.0	22.2				

Case Study "D"

This study is on a ⅜ in. x 0 coal from Greene County, Pa. It is from the Pittsburgh seam and was screened from the run-of-mine. It is an example of high-sulphur coal of which there are large reserves available in that area and which would make excellent metallurgical coking coal if the sulphur content could be reduced to a reasonable amount. The object of the testing was primarily one of sulphur reduction rather than ash.

Washability studies were made showing both ash and sulphur reduction. The Deister table was set to operate at a low gravity. Zone samples were taken along the table deck, with zones 1 through 6 being progressively along the coal discharge side of the table, zones 7, 8, and 9 along the refuse end with zone 7 at the diagonal corner. Table feed rate was estimated as 8.5 tph.

Table XI gives the screen, ash, and sulphur analyses of the table feed. The head ash of this coal is low, analyzing only 10.4 pct. The sulphur content is generally progressively higher in the finer sizes, increasing from a low of 2.30 in the +⅜ in. size to 2.98 pct in the 48x100-mesh and dropping slightly in the —100-mesh to 2.90 pct.

Since these analyses are "total" sulphur on a dry basis, it is important to know what the "inherent" or combined organic and sulphate sulphur content is for this coal, as no cleaning process will remove this "inherent" sulphur but only extraneous sulphur in the form of the iron sulphides of pyrite or marcasite.

In this particular test the exact "inherent" sulphur content was not available, but analyses of

Table IX. Screen Sizes and Ash Analysis of Clean Coal and Refuse from 3/16 in. x 0 Table Test, Case Study "C"

Size, Mesh	Clean Coal						Refuse					
	Direct, Pct		Cumulative, Pct				Direct, Pct		Cumulative, Pct			
	Wt	Ash	Wt	Ash	Wt	Ash	Wt	Ash	Wt	Ash	Wt	Ash
3/16x8	15.7	6.8	15.7	6.8	100.0	9.4	16.6	61.3	16.6	61.3	100.0	67.3
8x14	25.9	6.4	41.6	6.5	84.3	10.2	26.9	66.6	43.5	66.0	83.4	68.1
14x28	21.7	7.1	63.3	6.7	68.4	10.4	23.1	66.0	66.6	63.3	56.5	64.9
28x48	13.8	7.8	77.1	7.0	46.7	12.0	16.5	53.3	83.1	61.5	33.4	63.0
48x100	10.3	9.9	87.4	7.4	32.9	12.1	9.7	61.5	92.8	65.4	16.9	64.7
100x200	4.6	14.2	92.0	7.7	12.6	23.4	3.1	65.4	95.9	71.6	7.2	68.9
—200	8.0	28.7	100.0	9.4	8.0	28.7	4.1	71.6	100.0	67.3	4.1	71.6
Total	100.0	9.4					100.0	67.3				

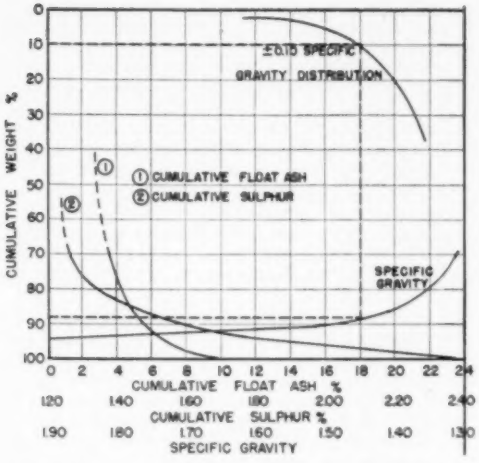

Fig. 14—Case study "D" washability of ⅜ in. x 100-mesh high-sulphur table feed.

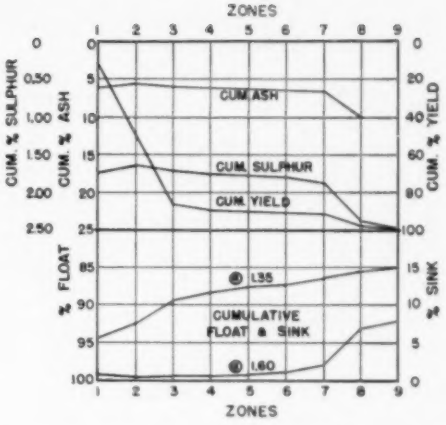

Fig. 15—Case study "D" ⅜ in. x 0 high-sulphur feed showing cumulative ash, yield, sulphur, float-and-sink byproduct zones.

other similar samples showed it to vary between 0.95 and 1.05 pct with possibly an average of 1.00 pct. One can assume, therefore, that if 100 pct of the extraneous sulphur were removed, the coal would still contain about 1.00 pct sulphur.

Fig. 14 shows the results of float-and-sink analyses of the table feed in the form of washability curves. From the viewpoint of ash removal, the coal is quite easy to clean. The ±0.10 sp gr distribution curve indicates that one can separate as low as 1.45 without running into difficult washing.

At this 1.45 sp gr a yield of 88.0 pct can be theoretically obtained, giving a cumulative float ash of 5.5 pct. The sulphur content of the 1.45 float coal is 1.52 pct. If it were possible to separate this coal at 1.35 sp gr, the sulphur in the float coal would coincidently be also 1.35 pct, assuming a perfect separation.

Table XII gives the yield, ash, and sulphur of the table product zones of case study "D". The lowest sulphur content of any of the zones is that of zone 2, which analyzed 1.64 pct. The cumulative product from zones 1 through 6 gave a total sulphur content of 1.80 pct with the combine refuse zones 7, 8, and 9 containing 11.89 pct sulphur.

From a table recovery standpoint it is interesting to note that 86.3 pct of the total table product came off the first three zones with a sulphur content of 1.72.

Tables XIII and XIV show the results of float-and-sink tests at 1.35 and 1.60 sp gr on the products from the various zones. In these tests the —100-mesh particles were previously screened out. Fig. 15 presents graphically the cumulative ash, sulphur, and yield percent and the cumulative float-and-sink at the two above-mentioned gravities, starting cumulatively with zone 1 and continuing through all nine zones.

As the washability curves in fig. 14 show, any separation below 1.45 sp gr would be quite difficult because of the high percentage of near gravity material which would, as expected, show up in the high percentage of sink in the table clean coal products at 1.35 sp gr as well as in the percentage of float coal in the refuse. At 1.60 sp gr the results,

Table X. Float-and-Sink Analysis 3/16x100-Mesh Products

Product	At 1.50 Sp Gr				At 1.60 Sp Gr			
	Float, Pct		Sink, Pct		Float, Pct		Sink, Pct	
	Wt	Ash	Wt	Ash	Wt	Ash	Wt	Ash
Clean coal	96.5	6.4	3.5	34.3	98.2	6.7	1.8	55.5
Refuse	4.7	18.6	94.3	69.8	8.5	21.2	91.5	71.0

Table XI. Screen Size, Ash and Sulphur Analyses of ⅜ in. x 0 High-Sulphur Feed, Case Study "D"

Size	Direct, Pct			Cumulative, Pct					
	Wt	Ash	Sul.	Wt	Ash	Sul.	Wt	Ash	Sul.
+ ⅜ in. rd	12.8	12.8	2.30	12.8	12.8	2.30	100.0	10.4	2.41
⅜x¼ in.	21.3	10.1	2.40	34.1	11.1	2.36	87.2	10.1	2.43
¼x14 mesh	37.2	9.5	2.31	71.3	10.3	2.33	65.9	10.1	2.44
14x48 mesh	19.6	9.8	2.46	90.9	10.2	2.36	28.7	10.8	2.61
48x100 mesh	3.4	11.7	2.98	94.3	10.2	2.38	9.1	12.9	2.93
—100 mesh	5.7	13.7	2.90	100.0	10.4	2.41	5.7	13.7	2.90
Total	100.0	10.4	2.41						

Table XII. Zone Analysis of Table Products from ⅜ in. x 0 High-Sulphur Feed, Case Study "D"

Zone	Direct, Pct			Cumulative Recovery, Pct			Cumulative Reject, Pct		
	Wt	Ash	Sul.	Wt	Ash	Sul.	Wt	Ash	Sul.
1	10.8	6.2	1.74	10.8	6.2	1.74	100.0	10.2	2.51
2	40.0	5.7	1.64	50.8	5.8	1.66	89.2	10.7	2.60
3	35.5	6.3	1.80	86.3	6.0	1.72	49.2	14.7	3.38
4	5.3	9.1	2.36	91.6	6.1	1.76	13.7	36.7	7.47
5	1.1	17.9	4.36	92.7	6.3	1.79	8.4	54.1	10.70
6	0.3	28.4	6.20	93.0	6.4	1.80	7.3	59.6	11.66
7	1.1	37.3	7.20	94.1	6.7	1.86	7.0	60.9	11.89
8	5.6	65.2	12.00	99.7	10.0	2.43	5.9	65.3	12.76
9	0.3	67.9	27.22	100.0	10.2	2.51	0.3	67.9	27.22
Total	100.0	10.2	2.51						

Table XIII. Float-and-Sink at 1.35 Sp Gr on Table Products from ⅜ in. x 0 High-Sulphur Feed,[a] Case Study "D"

Table Zone	Product Wt, Pct	Float, Pct			Sink, Pct		
		Wt	Ash	Sul.	Wt	Ash	Sul.
1	10.8	94.5	4.2	1.42	5.5	15.8	4.06
2	40.0	91.4	4.6	1.44	8.6	14.8	3.53
3	35.5	85.1	4.3	1.40	14.9	15.9	3.90
4	5.3	71.3	4.8	1.47	28.7	19.6	4.52
5	1.1	29.8	5.6	1.63	70.2	22.0	5.23
6	0.3	2.3			97.7		
7	1.1	2.4	5.5	1.79	97.6	37.2	6.97
8	5.6	2.2	5.6	1.73	97.8	66.6	12.07
9	0.3	0.0			100.0	26.3	26.05
Total	100.0						

[a] Float-and-sink analyses on +100-mesh sizes only.

as shown in table XIV, are excellent as far as the percentage of misplaced particles is concerned.

The minimum practical gravity of separation appears to be at 1.45. This is, of course, on an unclassified feed and with a fairly wide size range attempting to clean down to zero sizes. Better results could be achieved by closer sizing and also by making a middling or steam coal grade secondary product.

Table XIV. Float-and-Sink at 1.60 Sp Gr on Table Products from ⅜ in. x 0 High-Sulphur Feed,[a] Case Study "D"

Table Zone	Product Wt, Pct	Float, Pct			Sink, Pct		
		Wt	Ash	Sul.	Wt	Ash	Sul.
1	10.8	99.5	4.6	1.51	0.5	39.3	12.98
2	40.0	99.9	5.4	1.61	0.1	35.7	7.59
3	35.5	99.4	5.9	1.75	0.6	33.3	5.30
4	5.3	96.0	8.0	2.22	4.0	35.3	5.40
5	1.1	83.3	12.4	3.94	16.7	33.4	5.26
6	0.3	54.1	21.2	6.25	45.9	34.9	5.65
7	1.1	14.0	20.2	6.02	86.0	39.0	6.98
8	5.6	4.4	12.6	3.71	95.6	67.6	12.22
9	0.3				100.0	68.3	26.05
Total	100.0						

[a] Float-and-sink analyses on +100-mesh sizes only.

Summary and Conclusions

An attempt is made to discuss some of the theory of wet tabling and the phenomena surrounding tabling action. The Deister diagonal-deck table is described. In discussing the performance of this table in the cleaning of the finer sizes of bituminous coal, a study was made of four tests run on unclassified feeds, each test being made on different types and sizes of coals. The author has attempted to be objective in his analyses, presenting the facts of the different studies based on actual table performance.

The separation of different minerals on a wet concentrating table is the result of their differences in specific gravities, size and shape of particles, and the phenomena of stratification and hindered settling. These factors can be controlled and aided by many possible table adjustments, including speed, stroke, side tilt, end elevation, deck area, riffle design, shape of deck, and water distribution.

A diagonal-deck type of concentrating table offers the advantage of a more effective deck area at the section of the deck where it is most needed. For most coal washing a riffle design where the riffles are parallel to the differential head motion appears to give the most satisfactory results. A desirable table for wet washing or concentration of coal is one that provides the flexibility of adjustments so that it can be operated to fit the particular coal being treated and produce the desired results.

Capacities of tables will vary with their size, shape, and other operating characteristics, as well as with the particle sizes being treated, the inherent washability of the coal itself, and the results desired by the operator.

A study of the four cases incorporated in this report indicates that a high degree of performance was achieved under a wide variety of conditions. In comparing the results of any process, it must be remembered that the inherent characteristics of the coal must always be taken into consideration and that a true comparison between different processes can be made only when treating the same kind and size of coal.

It is believed that the type of coal washing table herein studied will produce a satisfactory product from fine sizes of coals and fulfill most requirements while operating on an unclassified feed. Test data indicate that bituminous coal fines of size consist ranges of ½, ⅜, ¼ and 3/16 in. x 0 can be treated effectively. If a higher degree of separation accuracy is required, a closely sized or classified feed may become necessary. This latter course may be especially desirable if the primary purpose is sulphur removal.

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Specific Safety Problems Applicable in West Virginia Mines

by Arch J. Alexander

This paper is a resume of a study undertaken by the West Virginia Department of Mines. The underlying and direct causes of accidents are determined in each occupational group. Then from this study, accident prevention courses, visual and written, are to be taken to the various working groups in the mines.

THE causes of accidents in West Virginia mines follow the main pattern of coal industry accidents in the United States—falls of roof, ribs and face, 56 pct; from haulage, 25 to 30 pct; blasting, 3 pct; electrocutions, 3 pct; explosions, 2 pct; and other causes, about 10 pct.

To arrive at some satisfactory conclusions regarding causes and prevention of mine accidents, the Department of Mines has made a study of all mine accidents, from 1945 to 1948, inclusive. During the last two years of this period, we did not have a major explosion, one that took the lives of as many as five men. The direct cause of these accidents is apparent in the reports but we who are making a close study into their frequency think that the virus lies much deeper, and we want to suggest an underlying cause.

The rapid growth of the coal industry in West Virginia, the largest producer of bituminous coal, has resulted in a demand for manpower which probably has never been equaled in mining history. At the turn of the century, West Virginia had 24,635 men employed in and about her mines, and they produced 21,153,341 tons of coal a year. In 1948, there were 125,669 men employed, and coal production for the year was 168,589,033 tons, an average increase of 35 pct for each year since 1900. Over 100,000,000 tons of this increase were mined in the southern counties of the state.

A safety analyst might read into the history of this development an underlying cause for accident occurrence. He might say that in such a vast, rapid development the available supply of trained miners became exhausted and that young, inexperienced men were employed in the mines by the thousands. Being unacquainted with the hazards of mining, it would be only natural for these young men to be injured or killed in too great a number. This assumption would not be borne out by the records. Men were not injured or killed in disproportionate numbers during their first years in a mine. The only reason we can give is that, during these years, they possessed an attribute of safety—*CAUTION*—to a much greater degree than in the years following.

The coal companies in West Virginia that have been operating over a number of years have a nucleus of employees who have been with them ten, twenty, thirty, and forty years; many of them having gained all of their mining experience with one company. Their habits of work have been

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steady and their proneness to accidents less than in any other group. They have remained long enough in one place to have acquired safety habits. When they make up a large percentage of the total employees of any one company, safety promotion is given a head start.

On the other hand, our accident figures indicate that there is a large segment of mine workers, especially in the rapidly developing fields, which is continually on the move. These miners have accumulated experience in two or more states and in the various fields of each state. Their restlessness is not psychologically conducive to the acquiring of safety habits.

They are of all ages, the young man in his twenties and the mine worker of sixty, and they claim experience in all the occupations in and about our mines. This roving band suffered an average of 42 pct of all our fatal accidents and as high as 60 pct in the counties having the highest accident rates.

The average for the group: Age, thirty-five; total mining experience, fifteen years; and length of service with the mine where they were killed, 4.7 months. They were injured or killed within the year not because of unfamiliarity with any new danger but because of their disregard for the old ones.

Our records show that a county with settled workers, many of whom are second and third generation miners, owning their own homes, and standing out as an integral part of their community, has a better safety record than a county in which there is a heavy labor turnover. In the counties having a heavy labor turnover, we see an answer to the problem by a breakdown of the accident records. In one of our largest producing southern counties, one fourth of the companies, large and small, mining one half of the production, have a safety record comparable to any group of mines in the nation despite their heavy labor turnover. Others could follow their example.

From this analysis, we also can see that the good worker is the safe worker. By the good worker, we mean the man who stays on the job year in and year out, who, in all probability, owns his home and is a steady, everyday man. He is the man who does not change companies too often, and every effort should be made to keep this type of worker at your particular mine as long as possible. This not only brings up the question of safety, but also a social question, which must be solved at each individual mine.

In West Virginia we have three well-known methods of mining, following plans that differ somewhat in pattern, but all with the same objective, and that is the extraction of the coal from the working faces. These methods of coal extraction have changed somewhat during this 4-yr study: Hand loading into mine cars, 1945, 42.7 pct; 1948, 31.7 pct. Mechanical loading by machines, 1945, 43.9 pct; 1948, 51.6 pct. Hand loading into conveyors, 1945, 13.4 pct; 1948, 16.7 pct.

Coal Loading by Hand into Mine Cars

We are experiencing a sharp decline each year in the amount of coal hand loaded into mine cars. However, this group of hand loaders still makes up the largest single group of men within our mines in a single occupation. They averaged, for the 4-yr period of study, 21.3 pct of all our mine workers, including supervisors. They made up 41 pct of all the men whose principal duties keep them at the working face. They still exceed the combined number of cutting machine, loading machine operators and helpers, and hand loaders into conveyors.

In this group of hand loaders into mine cars occurred 20 pct of our fatal accidents and the largest number of our serious nonfatal accidents. Even with such a showing, their record is better than the average for all groups, nine others proving more hazardous.

Of their total fatalities, 75 pct were from falls of roof, ribs or faces at the face. Being crushed against the face by cars, being caught in mountain bumps, and blasting, account for another 12 pct, making a total of 87 pct at the working faces. Of the accidents to these face men, 16 pct were from causes beyond their control, either carelessness on the part of a fellow worker or a hazard under the control of management and supervision. Few, if any, were not preventable.

Since most of these accidents, both fatal and nonfatal, to hand loaders were caused by the fall of roof, we immediately arrive at the conclusion that there is insufficient timber being set. This, without a question of a doubt, is the answer. Timber is not being set for one of three reasons; (1) The timber itself is not available, (2) the coal loader is not conscious of his danger, or (3) insufficient or ineffective supervision. Lack of discipline has been a great factor in accidents in our mines. Our records also show that the injuries or deaths of men in a working face were the result of insufficient timbering. Therefore, there can be only one conclusion—timber the working face and you will save the life and limb of the miners.

Machinemen, Cutting

Machinemen, cutting, come after brakemen in the number killed. So that we may keep together the groups with the most roof fall fatalities, we will discuss haulage later.

Machinemen account for only 5.25 pct of our total employees, but for 11.10 pct of the total fatalities. Of the fatalities, 68 pct were from falls at the face and 22 pct were from causes such as careless tramming, handling of the machine at the face, loose clothing, and being caught between machine and low top. Again, 10 pct were from causes beyond the control of the men.

Men cutting coal are within the group that are termed "skilled workers." They are a rather independent crew and, in most cases, work by themselves. Lack of training as to procedure in their work is the great killer of this group. Machinemen must be taught to reset timbers which they themselves knock out and to examine the roof before and during the cut. They must be taught to examine for gas before and during the process of cutting a place. However, examining for gas is secondary to inspection of the roof. Many a machineman knocks a timber down and never resets it. Again, a system of timbering at the face, carried out, even though many posts must be reset, will go a long way toward saving lives in this group.

Loading Machine Operators and Helpers

This group is very much like that of the machinemen and accounts for 4.95 pct of our employees and for 10.20 pct of the total number of men killed in our mines. Their record is only a little better than

that of brakemen and shuttle car operators. Eighty-three percent of the loading machine operators and helpers were caught under roof or rib falls at the face while loading coal, or on entries while loading slate. About 8 pct of them were killed by causes beyond their control.

Again, these men must be taught the dangers of unsupported roof. Since they are working around a great deal of noise, they cannot hear when the roof cracks or breaks. Therefore, we must timber for them. Loading machine operators should never be permitted to take a loading machine past the last crossbar or the last row of timbers, even though the machine must be stopped until the place is made safe by additional timbers. If this were carried out, coal production would be higher and a much better accident record would result.

Timber and Slate Men

Timber and slate men comprise 5.02 pct of our workmen and accounted for 7.60 pct of the total fatalities. Ninety percent of these fatalities were the result of falls of roof or rib, causes which were under their control. Six percent of these men were killed from causes beyond their control. A great many of the injuries and fatalities in this group could be avoided if the men would make themselves safe before starting to work. If they would set the necessary timber in the working faces or take the necessary precautions while cleaning up a slate fall, the fatalities would be reduced to nil.

Let me point out again the importance of supervision and discipline.

Hand Loaders into Conveyors

Men hand loading coal into either shaker or chain conveyors make up 8.12 pct of our total employees. They account for 6.9 pct of our fatalities. This is by far our safest group but, of the men killed loading conveyors, 77 pct of them were killed by the fall of roof and rib.

I believe the fine record made by hand loaders into conveyors is due to closer supervision. The average supervisor does not have too much territory to cover on a conveyor section because the coal is usually thin and a systematic method of timbering is carried out in most cases. The small percentage of men killed during conveyor loading may be used as a yardstick for all other occupations. Still, even with this fine record, there is room for vast improvement. A better method of safety timbering and better supervision would go a long way toward saving lives in this group.

Supervision

While supervisors are only 6.97 pct of our total employees, they account for 6.60 pct of our fatal accidents. Forty percent of the supervisors' deaths, fewer than in any other group, were from slate falls and 60 pct were from all other known causes. All of them could have been prevented and none took place in the saving of lives. The supervisor, traveling from place to place, constantly on the move, has a more dangerous job than almost any other worker within the mines.

In West Virginia we have an average of one supervisor for every eight workers and it seems that the leaders should set a better example for the men working under them by being more safety conscious themselves. I believe most face supervisors are far better production men than they are safety men. It is the way they have been taught. Now, if we intend to reduce hazards to this group, we must teach them the proper way to supervise as well as proper safety practices. This is strictly a local problem which must be worked out by the management of each mine.

Trackmen

Trackmen constitute 6.12 pct of our workmen and account for 4.20 pct of all fatalities. They can be credited with being, from a fatality standpoint, one of our safest groups of inside workers. Yet, 70 pct of their fatalities are the result of slate falls. There is no question that this group must be taught not to work in unsafe areas. This is a matter of roof testing and either keeping workers from dangerous places or seeing that they properly protect themselves. Again, 10 pct of the fatalities in this group were from causes beyond the control of the workers.

Drillers

Drillers, both coal and rock, are a small group accounting for 1.56 pct of our total workmen, but accounting for 2.04 pct of our fatal accidents. Slate falls at the face caused 46 pct of these accidents that could have been prevented. Thirty percent were from causes beyond the workers' control. Among these were accidents caused by runaway machines, runaway shuttle cars, the use of defective equipment, and loose clothing. Such accidents should never have happened. The drillmen at the face should take the necessary safety precautions of checking the roof and setting the necessary safety timbers. They should become aware of the other equipment being used near them.

Shot Firemen

This is one of the smallest groups in the mines, constituting only 1.65 pct of our total workmen. Yet, they had 2.30 pct of all the fatal accidents. Of these men, 83 pct were killed from falls at the face. Again, proper face precautions, the setting of safety timbers, and the awareness of roof fall dangers must be emphasized if we intend to save the lives of this group.

Haulage Crews

Brakemen: While all the groups in the mine are involved to a certain extent in haulage accidents, they occur mainly to our locomotive crews. Of all the workmen in a coal mine, brakemen have the most injuries. While accounting for only 5.25 pct of our total employees, about the same as cutters, they are charged with 12.50 pct of all underground fatalities. Their nonfatal injuries run very high in permanent disabilities and there are about eleven nonfatal accidents for every fatal accident. Fifty percent of these fatalities were caused by dangerous practices, such as riding front bumpers of cars or locomotives, riding between cars or locomotives, jumping in front of trips, and riding on the sides or top of locomotives and cars. All of these practices violate the rules of the Department of Mines. All of these brakemen were killed because they would not follow known safety rules.

Brakemen should be thoroughly schooled before they are permitted to get on a locomotive. A brakeman who continues to violate company rules, state rules and laws, etc., should be transferred from that particular job to another or discharged.

Twenty-seven percent of our brakemen were killed from causes beyond their control. These were by hazards which the supervisors or management should have discovered and removed. The remaining 23 pct were killed as a result of collisions, rerailing cars with a stiff jack, excessive speed, being caught in tight places and in thin coal, or by low roof over cars and locomotives. None of these are causes which cannot be corrected by the mine management. It is a well-known fact that haulage crews get the least supervision of any group of workers in a coal mine.

Motormen: Motormen constitute about the same number of the total employees as brakemen, 5.62 pct, but their fatalities are fewer than those of the brakemen, being 9.30 pct of all fatalities or about 3 pct less than the average for the brakeman. A great many of these fatalities to motormen are caused by slate falls, defective equipment, collisions, and runaways from trips other than their own. To reduce haulage accidents, which are a disgrace, there must be better control over haulage, the removal of dangerous entry roof, and an educating process for motormen and brakemen. Discipline enters the picture to a great extent and, in mines where discipline is somewhat lax, the danger of accidents to haulage crews is ever present.

Mechanized Mining

When mechanized mining was introduced in our state, we were told that the workers on the various mechanical units would be more concentrated, that they would have the benefits of instruction and supervision not possible for hand loaders into mine cars, and that safety would be advanced. To see if our expectations have been realized, we can look at the record:

	Tons Loaded Per Fatality	Man Hours Worked Per Fatality
Hand loaders into mine cars	1,070,607	823,450
Hand loaders into conveyors	1,318,064	906,381
Loading machine operators and helpers	2,780,041	373,694

As long as we measure our fatality rate on a tonnage-wide basis, the mechanical loading of coal, with the greater tonnage mined per fatality as compared with hand-loading methods, will present a better record. On the other hand, with the variables we now have in mining coal, an accident rate on a production basis does not express the actual safety record of a mine or of a company. Computing on the basis of man-hours exposure, the present high accident rate of loading machine operators and helpers shows that we have not improved the record but that it is somewhat worse.

The various principal occupations within a coal mine in their relative importance, as far as safety is concerned, and the accident record of each occupation have been presented. We are able to arrive at some definite conclusions very quickly.

The first conclusion is that there is a lack of roof support. About 60 pct of all mine fatalities are caused by roof falls. Therefore, if we are going to improve this situation, we must be sure that each working place has sufficient timber of the right length and that this timber is set. When we accomplish this, we will have gone a long way toward cutting down fatal accidents in the fall of roof.

In the past year, there has come into vogue a new system of roof support in coal mines and that is roof bolting. This system controls the roof either by bridging or by holding the roof directly to an above strata. At the present time, we have some 103 mines in West Virginia using this system of timbering successfully, and there is no question in our minds that roof bolting has a great future. Like every other system of roof support, it must be properly studied so that the right equipment can be used, a definite system set up as to its use, and that system carried out. We are gathering information on this subject and are encouraging the use of bolts under a controlled system. We believe that roof bolting is the most progressive step that has been made to date in coal mining. However, as a system of roof support, it is no better than the people who use it.

Supervision is one of the weakest links in our chain of mining. We select a man as foreman because he happens to be an expert motorman, machineman, or outstanding man in his line of work. We never think, generally speaking, that this man needs special instruction in the many things over which he has supervision. The first step in any accident prevention program must start with the supervisor. This instruction takes time and costs money, but we believe it is the only way that any mine can improve its accident record.

Many years ago, when mechanized mining was first started, it failed in a good many places. Our ignorance of the job caused this failure. It did not take us long to study each step of the work or cycle in mechanized mining. After we had made this study, we trained our men, and the results were very gratifying.

The West Virginia Department of Mines, profiting by this experience, has been studying mine accidents by occupation. For instance, we have broken down mine accidents as to timbermen, trackmen, haulage crews, and machinemen; in fact, every category of underground work, to see what, besides roof, was killing our men. The results of this study were amazing. For example, we found that there were only 11 ways that 98 pct of the trackmen in the mines of West Virginia were being killed. There were even fewer ways in other mining occupations.

It is our intention to set up an accident prevention course that can be taught very rapidly. It is planned to present in one 2-hr session a course on accident prevention to each group of workers, one for trackmen, one for timbermen, etc. We will give this course in simple language. We intend to illustrate with slides, and, when a man has finished his 2-hr course, he will be given a booklet covering the subject matter of the course.

This will be a slow process since, under our present setup, it is not probable that we can reach more than 100 mines a year, and, in West Virginia, we have some 2600 coal mines. It is our intention to keep a close record on the mines where this instruction has been given in order to determine whether the instruction has been worthwhile. We believe that we will find that it has been.

In conclusion, let me say that we believe that it will be necessary that the top bracket of management be interested in safety and in seeing that the safety programs, as set up by their company, are carried out. If top management is not sold on safety, there is very little chance for safety in coal mines.

Air Pollution by Industrial Fumes, Gases, and Dusts

by Louis C. McCabe

The control of dusts and fumes of submicron size is involved in many process industries. This paper presents in tabular form the quantitative data from a number of metallurgical operations and discusses allowable discharge in terms of effluent-weight ratio. Typical installations for recovery of stack effluents are cited.

THERE are many sources of smoke, fumes, gases, and dusts that may cause undesirable contamination in an industrial area. Fumes and gases are the result of chemical action or volatilization, while dusts are produced in processes of disintegration and dispersion. Commonly, smokes are suspended particles formed by combustion or destructive distillation. Methods for controlling smoke are generally known, and great improvements have been made in equipment for removing fine dusts and fumes from gases.

In most of the industrial centers of the country, the chief duty of the air-pollution abatement agency is the elimination or reduction of emissions resulting from improper combustion. Correct methods of firing, proper fuel selection, and installation of properly designed equipment may accomplish this. It is becoming more widely recognized that much of the contamination of an industrial atmosphere is caused by dusts, fumes, and gases that are not related to the process of combustion. As such emissions are usually a functional part of plant operation, they rarely can be prevented even by better design and careful operation. When dusts, fumes, and gases are an integral part of the operation, mechanical equipment must be designed to remove these emissions.

Most of the dusts and fumes that should be controlled are less than 1 micron in size. Control of such small particles is largely a new and untried field in many industrial areas. However, where conditions have been critical, effective control equipment for dusts, fumes, and gases has been in operation for many years. The problem is not one of smoke abatement and requires an entirely different set of standards for effective solution. Identical standards need not be applied to all industrial areas because of differences in meteorology, physiography, plant location, plant density, and nature of stack effluents. In this paper, discussion of control of atmospheric contamination by industrial plants refers specifically to the Los Angeles area, but the method of approach is applicable to any area where similar problems are encountered.

Quantities of particulate matter reported in analysis of stack effluents indicate the need for standards of emission. Many types of dusts and fumes that contribute materially to the air-pollution problem are discharged into the atmosphere at a lower rate than 0.40 grain per cu ft at standard conditions. On the other hand, some stacks discharge at a higher

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Table I. Summary of Data

Type of Process	Process Weight, Lb per Hr	Process Time	Stack Loading Grain per Cu Ft at Std.		Effluent to Weight Ratio, Cu Ft at Std.	Hourly Max Rate of Discharge, Pct	Average Max Rate of Discharge, Pct
			Avg	Max per Hr			
Red brass	500	72 min	0.250	0.393	273	1.53	0.97
Red brass	435	84 min	0.254	0.395	289	1.63	1.04
Red brass	1,500	152 min	0.320	0.559	172	1.38	0.79
Red brass	1,620	130 min	0.595	0.903	147	1.92	1.26
Yellow brass	470	102 min	0.410	0.780	231	2.57	1.35
Yellow brass	535	90 min	0.452	0.762	197	2.15	1.27
Brass smelting	2,000	25 hr		0.390	226	2.11[a]	1.26
Brass smelting	630	95 min		0.985	282	3.95	
Lead smelting	5,000	24 hr		0.355	355	3.01[a]	1.80
Steel, open hearth	14,500	8 hr	0.798	2.000	55.6	1.58	0.63
Steel, open hearth	10,200	10 hr	0.863	1.720	53.1	1.31	0.65
Gray iron	9,250	80 min	1.604	1.604	32.5	0.75	0.75
Gray iron	13,450	156 min	1.110	1.110	91.5	1.45	1.45
Gray iron	18,900	320 min	0.798	0.798	49.5	0.56	0.56
Gray iron	8,350	Data by outside laboratory				1.07	1.07
		Gray iron discharges are constant throughout melt cycle. Average and hourly rate of discharge are equal.					
Steel, electric	3,325	130 min	0.306	1.17		1.02	0.29

[a] Pct, calculated, based on average increase found in the nonferrous industry.

rate, but because of the coarseness of the material the nuisance created is limited to the immediate neighborhood and perhaps only to the property of the company concerned. Much of the characteristic blue haze, which contaminates the atmosphere and contributes to low visibility, is due to particle sizes near the low range of the wave length of visible light, that is, 0.3 to 0.5 micron diam.

It is not possible to separate the industries into well-defined groups according to the physical state of their stack discharges, but for the purpose of this discussion they are grouped as follows: Dust: metallurgical, clay and rock products. Fumes: food processing, fertilizer, paint, roofing and asphalt. Gases: refining, chemical.

Dusts and fumes standards now in effect in Los Angeles County were developed after a year's study of the stack discharges of representative plants. Three or four furnaces are represented in each type of metallurgical operation chosen for basic information. The data (table I) are representative of four types of metallurgical industry, namely, the nonferrous, steel, gray iron, and electric iron. The nonferrous operations comprise the largest group numerically, but in gross tonnage of processed material the combined steel industry is larger.

The development of these standards[1] has been discussed elsewhere; significant factors in their use are:

1. Effluent-Weight Ratio—This is the total volume of effluent in cubic feet at standard conditions per pound of material processed and indicates the average dilution ratio for each plant in a given industry. Throughout each industry this ratio tends to hold constant.

2. Hourly Mass Rate of Emission—This is the maximum discharge of solid material occurring in any 1-hr interval expressed as a percentage of the total hourly process weight.

3. Average Mass Rate of Emission—This is the average discharge for the entire process cycle expressed as a percentage of the total process weight per cycle.

Maximum stack loadings, in grains per cubic foot at standard conditions, range from 0.393 in a red-brass operation to 2.0 in a steel open hearth. The maximum allowable discharge, expressed in pounds per hour for selected weights, is presented in tabular form in table II. This was done at 50-lb intervals up to 20,000 lb per hr and in 10,000-lb intervals up to 50,000 lb per hr. A further restriction on the allowable discharge was made in that it becomes constant when 40 lb per hr is reached, regardless of the process weight involved. The collection efficiencies used were those that can be obtained with available equipment within the economic reach of industry. It was felt the additional cost of high-efficiency collection equipment can be carried better by large industries; the smaller industries will be required, therefore, to operate their collecting equipment at 80 pct efficiency and the large industries to operate with collection efficiencies of approximately 90 pct.

Open-Hearth Steel: Two pilot installations on the open-hearth steel furnaces were completed during the year: (1) a venturi scrubber and (2) an electrostatic precipitator. The pilot tests proved that both of these types of equipment will meet Los Angeles standards successfully in the collection of fume and dust from the open-hearth furnaces. Economics, involving water and power consumption, are important factors in the system selected.

[1] L. C. McCabe, A. H. Rose, W. J. Hamming, and F. H. Viets: *Ind. and Eng. Chem.* (Nov. 1949) 41, No. 10.

Operating data from one open-hearth is as follows:

a.	Average metal charge	129,498 lb
b.	Average lime charge	5,620 lb
c.	Average coke charge	1,450 lb
d.	Average total charge, Jan. 1 to June 1,1949	136,568 lb
e.	Average heating cycle (start of charge to finish tap)	8.083 hr
f.	Average charge per hr	16,900 lb
g.	Combination firing	30,000 cfh natural gas and 72 gph fuel oil
h.	100 pct oil firing	250 gph fuel oil used when on gas curtailment
i.	Volume of flue gases	12,500 to 14,000 cfm at standard conditions

The temperature of the gases entering the stack was approximately 1300°F.

The dust loading rate in the stack gases was at the maximum, 2.00 grains per cu ft at standard conditions; the charging average was 0.87 grain per cu ft; and the melt-down average was 0.51 grain per cu ft.

The average charge per cycle for the second open-hearth plant was 111,000 lb on a 10.25-hr cycle. The temperature entering the stack ranged from 1100° to 1250°F. The maximum dust loading was 1.44 grains per cu ft and the minimum, 0.08 grain per standard cu ft. The average was 0.60 grain per standard cu ft.

Electric Steel: The fume-discharge problem of the electric steel furnace was solved by using a hood system, which utilized a limited amount of air and discharged through a wet dynamic precipitator. Operating installations of this type of equipment were placed in use during the year, and tests proved these units to be most satisfactory.

Gray Iron: This division of the metallurgical industry is farthest from a solution of its problem. Tests in cooperation with the Air Pollution Control District were made in an effort to determine characteristics of the effluent discharges from the cupolas. As a result of these tests, the District believes that this problem will be solved through the application of equipment similar to that employed in the steel industry.

Nonferrous Industry: Three types of collection equipment have been tried with varying success on the submicron fume discharge encountered in this industry. Modified wet scrubbers and mechanical separators were discarded as unsatisfactory. Filter-type collectors, however, proved satisfactory when proper hooding was employed. Changes in operating techniques were also tried with limited success. Furnaces were slagged throughout the entire operating cycle, including the pouring phase. On alloys of moderately low zinc and lead content, this method was most satisfactory. As the lead and zinc content increases, the slagging technique appears to be less effective. It is probable that collection equipment will be required in the high zinc alloy operations.

Special air-pollution control methods are required in a wide variety of industries such as rendering plants; chemical fertilizer plants; chromate manufacturing; varnish, paint, and resin plants; detergents; vegetable-oil processing; oil refining, sulphur production; acid plants, paper coating; and food processing.

Table II. Maximum Allowable Discharge per Hour

Process Wt per Hr (Lb)	Allowable Disch. per Hr (Lb)	Process Wt per Hr (Lb)	Allowable Disch. per Hr (Lb)	Process Wt per Hr (Lb)	Allowable Disch. per Hr (Lb)	Process Wt per Hr (Lb)	Allowable Disch. per Hr (Lb)
50	0.24	1,300	3.26	3,500	5.82	8,500	9.03
100	0.46	1,400	3.40	3,600	5.91	9,000	9.36
150	0.66	1,500	3.54	3,700	5.99	9,500	9.67
200	0.852	1,600	3.66	3,800	5.77	10,000	10.0
250	1.03	1,700	3.79	3,900	5.85	11,000	10.63
300	1.20	1,800	3.91	4,000	5.93	12,000	11.28
350	1.35	1,900	4.03	4,100	6.01	13,000	11.89
400	1.50	2,000	4.14	4,200	6.08	14,000	12.50
450	1.63	2,100	4.24	4,300	6.15	15,000	13.13
500	1.77	2,200	4.34	4,400	6.22	16,000	13.74
550	1.89	2,300	4.44	4,500	6.30	17,000	14.36
600	2.01	2,400	4.55	4,600	6.37	18,000	14.97
650	2.12	2,500	4.64	4,700	6.45	19,000	15.56
700	2.24	2,600	4.74	4,800	6.52	20,000	16.19
750	2.34	2,700	4.84	4,900	6.60	30,000	22.22
800	2.43	2,800	4.92	5,000	6.67	40,000	28.3
850	2.53	2,900	5.02	5,500	7.03	50,000	34.3
900	2.62	3,000	5.10	6,000	7.37		
950	2.72	3,100	5.18	6,500	7.71		
1,000	2.80	3,200	5.27	7,000	8.05		
1,100	2.97	3,300	5.36	7,500	8.39		
1,200	3.12	3,400	5.44	8,000	8.71		

Where the process weight falls between the figures stated, the values of allowable discharge per hour will be interpolated. In no case will particulate matter in excess of 40 lb per hr be allowed to emit from any one source.

Process weight shall be defined as the total weight of raw materials or materials entering the process net weight of finished product. Solid fuels charged will be considered as part of the process weight but liquid and gaseous fuels and combustion air will not.

In continuous operation, the average rate of feed shall be used.

In batch operation, the total batch weight divided by the operating time of a batch cycle shall determine the process weight per hour.

In the rendering industry, as in others where odors are the chief air-pollution problem, much progress has been made. Among the methods used for odor reduction were incineration, wet scrubbing, and chemical treatment by proprietary formulas. Incineration is proving to be by far the most successful and practical method. Removal of odors by wet scrubbing has been used successfully in the ammonia-odor nuisance but is generally limited to odors of a water-soluble nature. The method has been used also in the varnish cooking industry. An experimental installation was made by one rendering plant for neutralizing odors by means of a chemical additive in the effluent gases. This proved to be unsuccessful and was abandoned in favor of incineration.

During 1949, the Hancock Chemical Co. built a plant at Long Beach for converting hydrogen sulphide to sulphur. This plant is presently producing 50 tons of sulphur from H_2S delivered by three refineries and is expanding its facilities so that it can process hydrogen sulphide from three additional major refineries. Prior to construction of the Hancock plant, this hydrogen sulphide was burned as fuel along with the other refinery gases, and the sulphur dioxide produced was released to the atmosphere.

Some pilot-plant work is being done on ultrasonic methods of agglomerating and recovering dusts and mists. Fume problems in the coffee-roasting industry can best be solved by incineration. One installation has been made to test the design factors used in construction.

Drying Fine Coal in the Entrained and Fluidized State

by V. F. Parry and E. O. Wagner

This paper summarizes investigations during 1949 on three pilot plants for drying low-rank fine coal by entrainment in hot gases. Detailed operating results on processing seven coals having moisture ranging from 24 to 62 pct, formulas and data for calculating performance of large units, and general conclusions on the problem of drying fine coal are presented.

THIS paper presents a summary of investigations by the Coal Branch of the Bureau of Mines during 1949 on the removal of internal moisture from low-rank coals by processing them in the entrained and fluidized state. A paper on this subject, giving a summary of the early work, was presented at the 1949 AIME annual meeting[1] and gave a theoretical discussion of the time and heat required to dry various sizes of low-rank coal. The investigations on two pilot plants were presented in the foregoing paper. During 1949, three additional pilot plants were built to study phases of the drying problem and to investigate different techniques of handling fine coal in the entrained and fluidized state. The present paper gives operating data on the new units and presents detailed data on the drying of seven coals having bed moisture ranging from 62 to 24 pct.

[1] V. F. Parry, J. B. Goodman, and E. O. Wagner: Drying Low-Rank Coals in the Entrained and Fluidized State. *Trans.* AIME 184, 89-98; *Min. Eng.*, April 1949.

The problem of removing surface moisture from fine coal is principally one of heat exchange and dispersion of the coal particles so that heat can reach the liquid moisture to permit evaporation, and it does not involve the factors of size and time of contact to any great extent. If enough heat can reach the surface of the particles at temperatures higher than 250°F, moisture evaporates almost instantly, and the factors of time and size are of secondary importance compared with the factors of heat balance and dispersion of the coal. Therefore, it does not appear to be necessary to employ a fluidized bed when removing surface moisture, and the most efficient process would be one in which the coal particles are entrained in a hot gas stream of optimum temperature traveling at a velocity that causes the particles to disperse into a low-density phase.

When low-rank coals are dried, the fluidized bed is necessary to provide control of time of contact. This is done by adjusting the superficial velocity which controls the density or dispersion of the particles. The density of beds having fluidized properties suitable for drying low-rank coals appears to range from about 5 up to about 15 lb per cu ft. When the density is below 5 lb per cu ft, which represents a low-density phase, optimum conditions for removal of surface moisture should be found, and this requires superficial velocities of about 15 fps.

The problem of removing bed moisture from low-rank coals involves consideration of size of coal, temperature, mass flow, heat balance, pressure, time of contact, and method of dispersing the coal in the heating mediums. The experimental work presented here gives data on these factors. Coal sizes up to ½ in. x 0 were dried with hot gases having initial temperatures ranging from 1900° to 2400°F. Coal

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was fed pneumatically and by screw conveyors, and the time of contact in the drying column ranged from 19 to 230 sec.

The principal object of the experimental work cited in this report was to ascertain the relationship of the various factors affecting the drying of fine low-rank coals containing various amounts of natural bed moisture. A few experiments were made on the removal of surface moisture, but the results are not presented as they were not conclusive, and additional work on optimum temperature of hot gases is necessary before results on this problem can be reported.

Summary and Conclusions

This report presents results of experimental work on the fluidized drying of low-rank coals from Colorado, Wyoming, Alaska, Texas, and Greece, which contain 24 to 62 pct natural bed moisture. They were processed in three pilot plants to find optimum conditions for removing 90 to 95 pct of the moisture. The factors of temperature, time of contact, heat balance, and size were studied, and two methods of dispersing the fine coal in the fluid stream were employed.

The fluidized process for drying fine coal has been under investigation for about two years by the Coal Branch of the Bureau of Mines at Denver, Colo. A preliminary report on the work was presented at the 1949 annual meeting of the AIME. The main features of the drying process are described as follows:

Hot gas under moderate pressure is generated in a combustion chamber by the combustion of pulverized coal or gas. The temperature of the hot gases issuing from the burner is regulated by recirculating inert products of combustion to reduce the free-oxygen content and to produce a temperature suitable for drying the coal. This hot gas is jetted at velocities of about 80 fps into a drying column or heat-transfer zone where it contacts the fine coal and creates a fluidized or entrained mixture of gas and coal in which heat exchange occurs rapidly to evaporate the moisture. The mixture of hot gases and fine coal moves upward in an entrained and fluidized condition to suitable cyclone separators, where the dried coal is deposited and the moisture vapor is discharged. Part of the discharged gases is recirculated to control the temperature and velocity of the gases in the drying column.

This process causes extremely rapid rates of heat interchange between the hot gases and coal particles and results in maximum thermal efficiency. The time required to pass the coal through the system ranges from about 20 to 230 sec, depending upon the size of the coal treated.

Experimental work on various sizes and ranks of coal dried under various conditions in three pilot plants supports the following general conclusions:

1. The capacity of a fluidized drying unit is expressed approximately by the following empirical formula, which shows the relationship of the major variables:

$$C = \frac{3.2ST_1^{0.56}}{M_1}$$

where C = lb of raw coal per hr per sq ft,
S = mean superficial linear velocity of the hot gas and vapors in the column at the temperature and pressure of the system, ft per sec,
T_1 = temperature of hot gases jetted into the raw coal, °F,
M_1 = moisture content of the raw coal, pct.

The foregoing formula was derived for a mixture temperature of 300°F and it assumes that 93 pct of the moisture in the coal is removed. The capacity of a drying column is directly proportional to the linear velocity and nearly directly proportional to the temperature of the hot gases while being inversely proportional to the moisture content of the coal.

2. For the average range of coals tested, the practical capacity of a drying unit is about 500 lb of moisture liberated per hr per sq ft. This means that if 20 pct of moisture is liberated from a coal, the capacity of the unit should be about 2500 lb per hr per sq ft, whereas if 35 pct moisture is liberated, the capacity will drop to about 1430 lb per hr per sq ft of drying-column area.

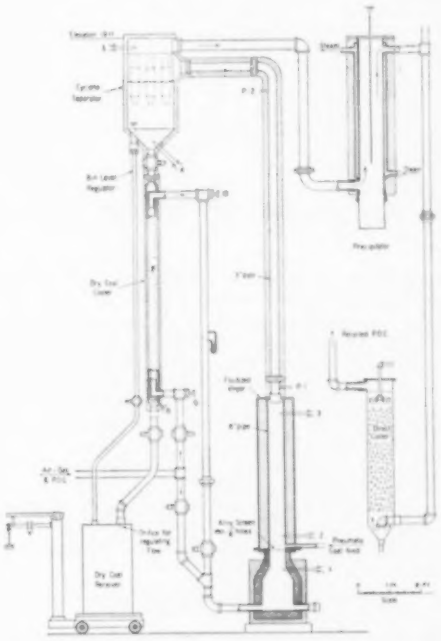

Fig. 1—No. 3 pilot plant for drying low-rank coal in fluidized state, Denver, Colo.

3. The efficiency and capacity of the fluidized drying system increase in almost direct proportion to the hot gas temperature, however, the practical limit of temperature is around 2000°F because temperatures higher than 2000°F promote the formation of ash deposits near the jets. Temperatures as high as 2500°F were studied with good operation except for the formation of ash deposits.

4. Coals having high bed moisture can be jetted with high-temperature gases with virtually no effect on the coal substance when 90 pct of the moisture is liberated. When all of the moisture is liberated,

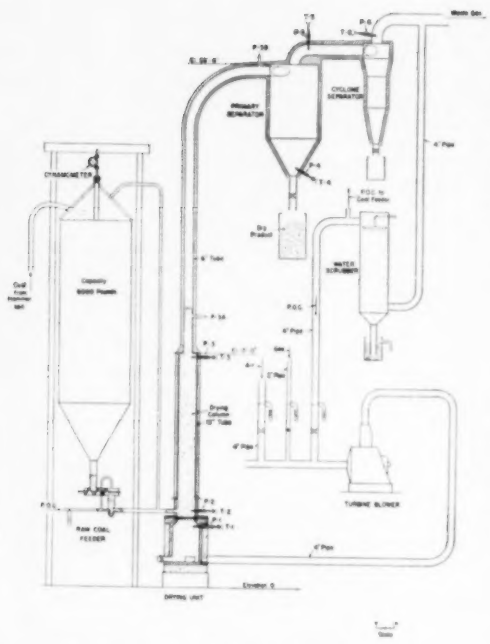

Fig. 2—No. 4 pilot plant for drying fine coal, Golden, Colo. Capacity 2400 lb per hr.

some devolatilization of the coal substance occurs, depending upon the temperature of the mixture.

5. The thermal efficiency of the fluidized drying process is 90 to 93 pct, excluding radiation and with no recovery of heat from the dried coal. In the small pilot plants, the overall thermal efficiency ranged from 73 to 85 pct because of the considerable radiation. However, in a large drying unit, the overall efficiency should approach 90 pct.

6. The density of the coal and gas mixture in the drying column is controlled by the superficial velocity and the size distribution of the coal particles. With equal velocity, the density of the mixture will increase with the average size of coal treated and this increases the time of contact.

7. Considerable degradation of the coal particles, caused by rapid evolution of moisture and shrinkage of the coal, occurs in the drying column. The measured degradation in average size ranged from 15 to 81 pct when Alaskan subbituminous coals and Greek lignites were treated. In general, the amount of degradation approximately equals the bed moisture of the coal treated, but the size of the particles and physical structure of the coal have some bearing on degradation.

8. It has been demonstrated that it is feasible to dry any coal completely and to effect devolatilization of the coal in the fluidized system. The amount of heat transmitted to the coal particles governs the extent of drying or devolatilization. There appears, however, to be an optimum point in removing bed moisture, taking into account capacity and estimated cost of removing moisture. A disproportionately longer time is required to remove the last traces of moisture, and the capacity decreases. It appears feasible to reduce the moisture in any coal to 2 to 5 pct while attaining maximum capacity and efficiency.

9. The size of the coal particle automatically regulates the time it will remain in the fluid system—the smallest dust particles evidently move with the gas at the same superficial velocity and, as the particle increases in size, it is carried less rapidly by the gas owing to its increased weight and drag. This conclusion is indicated from measurements of pressure drop and the residual moisture in different sizes of treated coal. The size distribution of the coal particles also affects operation of the fluid bed. A preponderance of large pieces will clog the column, and it seems necessary to float the larger pieces with adequate fine material.

10. The drying of a given coal to any degree is a straightforward heat-balance problem. The heat required to evaporate the moisture and raise its temperature to the mixture temperature plus the sensible heat deposited in the dried coal up to the mixture temperature is balanced against the sensible heat of the drying gas between the high and mixture temperatures. This gives the heat balance of materials excluding radiation. The extra sensible heat from the hot gas to supply radiation is then calculated. The unit quantity of materials thus calculated is then fed to the system at predetermined rates to establish the desired superficial velocity.

11. The dried coal dusts are free flowing because of the low moisture content. Although the compacted bulk density ranges from 45 to 56 lb per cu ft, the normal angle of repose of the coal is approximately 45°. The dried dusts flow through orifices by gravity at rates similar to the flow of liquids; for example, the flow of dust, in pounds per minute, equals $14.5\ D^{2.65}$, where D is the diameter of the orifice in inches. When the dust is discharged through a 7-ft long pipe, the rate of discharge is approximately four times greater than through an orifice of equal diameter.

12. It is estimated that the investment cost of fluidized drying units should be in the range of $150 to $200 per ton of daily capacity, depending upon the moisture content of the coal. The cost of drying a lignite to remove 670 lb of moisture per ton is estimated at $0.286 per ton. It is indicated that the unit cost of removing moisture should range from $0.85 to $1.00 per ton of water removed.

Experimental Work, Description of Pilot Plants

The No. 3 pilot plant is shown in fig. 1. The object in building this unit was to prepare samples of dried coal for briquetting tests and to study some of the problems of handling and cooling of dried fine coal. In this unit, coal was fed pneumatically to the drying column from a weighing and charging system described in the previous paper. The separating cyclones were about 19 ft above the ground level and connected to a dry-coal cooler through which the hot, dried coal moved to the coal receiver. The mixture of gas and air going to the combustion chamber circulated through the cooler. In the cyclone separator, three small cyclones were used for dust separation. This vessel was insulated to avoid condensation of steam, which had a concentration of about 50 pct in the waste gases.

The fine dust in the waste gas passing the primary cyclone separators was sent through an electrostatic precipitator. This unit did not function very satisfactorily because of shorting out caused by condensation, but when operating without shorting, the

collection of dust was satisfactory. Part of the waste gas was drawn back through a water scrubber to supply recycled products of combustion for the heating system.

A larger drying unit was constructed to study several factors of operation and to treat large batches of coal. This No. 4 plant is shown in fig. 2. The basic principles of operation employed in other units were incorporated in this plant, but the capacity was quadrupled, and provisions were made to gain further experience in handling materials.

The raw crushed coal was stored in a large cylinder suspended on a dynamometer. A screw feeder driven by a variable-speed motor was attached to the cone of the cylinder and transferred coal into the line feeding the drying unit. A jet of products of combustion moved the fine coal through the 1¼-in. id conduit to the drying column. The distance between the feeder and the dryer was about 30 ft. This coal feeder handled ¼-in. coal at any desired rate up to 3000 lb per hr, but the transfer conduit and the carrying gas were adjusted to provide 1 to 2 cu ft of gas per lb of coal at a velocity of about 75 ft per sec.

Table I. Calculation of Drier Operation

Test No.	21	30-B	32-A
Moisture to be removed from coal, pct, (D)	95	95	99
Moisture in raw coal, pct, (M_1)	58.8	62	37.6
Moisture in dried coal, pct calc., (M_2)	6.7	7.1	5.7
Improvement ratio $(100 - M_2) \div (100 - M_1)$, ($R$)	2.263	2.442	1.512
Moisture removed per lb of coal, $= (1 - 1/R)$, (X)	0.558	0.59	0.338
Dried coal per lb of raw coal, $(1 - X)$	0.442	0.41	0.662
Temperature of combustion gases (developed), °F, (T_1)	2,250	2,100	2,100
Temperature of combustion gases (theoretical flame), [a]	2,500	2,400	2,400
Temperature at top of drying column, °F, (T_2)	300	275	250
Temperature of dried coal, °F, (T_3)	275	250	225
Superficial velocity in column, ft per sec, at T_2 and P, (S)	15	10	10
Heat required to dry coal, Btu per lb:			
Latent heat in moisture evap. $= 1058X$	591	624	358
Sensible heat in steam, 60°F to T_2	61	52	28
Sensible heat in dried coal, 60°F to T_3	24	19	27
Total heat required excluding radiation, (H_1)	676	695	413
Sensible heat in POC[b] at T_1, Btu per cu ft (from chart)	48.2	44.5	44.5
Sensible heat in POC at T_3, Btu per cu ft	5.0	4.2	4.0
Heat transferred to coal, Btu per cu ft, (H_2)	43.2	40.3	40.5
POC required $= H_1/H_2$, cu ft per lb of coal	15.7	17.2	10.2
Volume of moisture liberated $= 21X$	11.7	12.4	7.1
Total gas and vapor in column, cu ft per lb coal, (V_1)	27.4	29.6	17.3
Volume of total gas at T_2 and P, cu ft per lb coal, (V_2)	48.1	50.0	28.3
Dryer capacity, lb per hr per sq ft $= S \times 3600/V_2$	1,122	720	1,270
Gas required:			
Overall efficiency of drier, assumed, (E)	0.85	0.75	0.75
Net heat required $= H_1/E$ Btu per lb coal, (H_3)	795	927	510
Cu ft of gas per lb of coal H_3 per 880	0.904	1.053	0.580
Air required = 10 x gas cu ft per lb	9.04	10.53	5.80
POC recirculated (from combustion chart)[c]	4.20	5.78	3.19

[a] The theoretical flame temperature is 200° to 350°F higher than the developed temperature, depending upon the design of the combustion chamber.
[b] Products of combustion.
[c] A combustion chart is calculated for gas or coal showing the ratio of air POC vs. theoretical flame temperature when burning with 10 pct excess air.

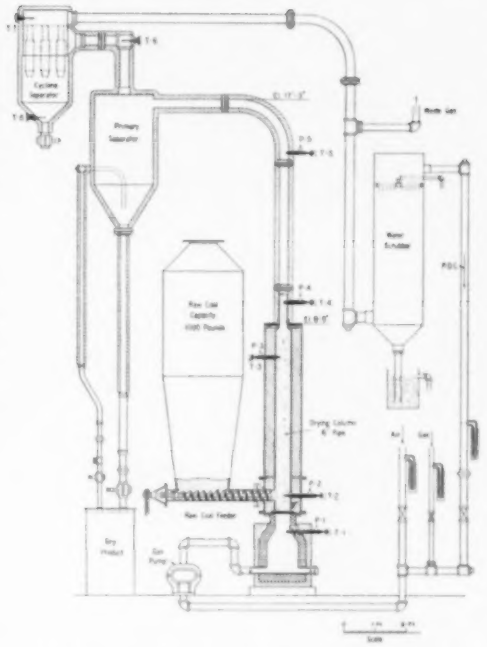

Fig. 3—No. 5 pilot plant for drying coal, Golden, Colo.

In the No. 4 plant, a turbo blower was used to feed the gas and air to the combustion chamber. This introduced the factor of constant pressure on the system, which caused a little trouble. The pressure drop or resistance through the drying column varies somewhat, and with constant pressure available, the flow will vary with the resistance, resulting in pulsations during operation. However, the No. 4 unit performed very well, particularly on the light-density fluid beds.

In this plant a primary separator consisting of a plain cylinder with tangential inlet and center off-take was installed. About 99 pct of the dried product was caught in this separator with little pressure drop. The off-gas passed through a single cyclone separator where fine dust, down to about 10 microns, was separated with a pressure drop of 4 to 6 in. of water.

The grid above the combustion chamber of the No. 4 plant was constructed of plastic refractory approximately 1¼ in. thick with twenty-nine ⅝-in. holes cast in a circular pattern. The lower 12 in. of the drying column was lined with refractory, but the upper section was a plain 12-in. steel tube insulated on the outside.

The No. 5 pilot plant, illustrated in fig. 3, was constructed to study the factors of size and degradation and also to provide a simple unit for testing various coals. The drying column and combustion chamber used in plant No. 3 were incorporated in this unit, the feeding system was changed to a screw feed from a closed hopper, and a simple primary separator similar to the one employed in the No. 4 plant was included. The perforated grid above the combustion chamber was made of ¼-in. alloy plate drilled with thirty-six 17/64-in. holes. A constant-

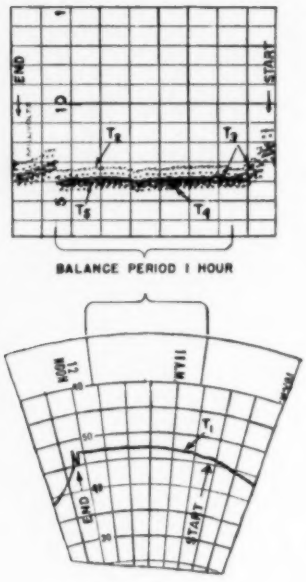

Fig. 4—Temperature data for test 30-B, pilot plant No. 5.

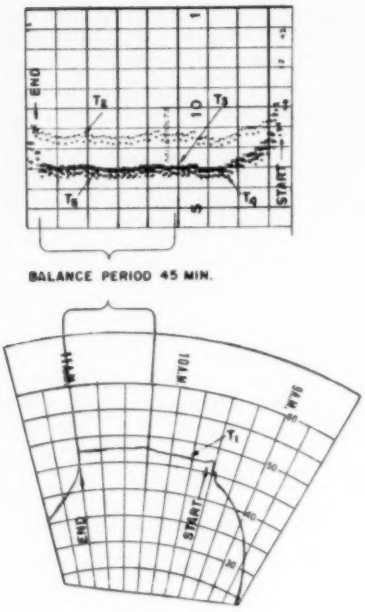

Fig. 5—Temperature data for test 29, pilot plant No. 5.

volume gas pump was used for introducing the gas and air mixtures into the combustion chamber under pressure.

The No. 5 plant has advantages for testing the drying characteristics of small batches of coal up to 1000 lb and is useful for research on the several factors affecting drying. Its operation is quite simple.

Operating Procedure: The procedure in making tests is approximately the same on each of the pilot plants and is described as follows with respect to plant 5, fig. 3:

The coal to be dried is prepared by crushing in a hammermill to the desired size. For routine tests, ¼ in. x 0 size is employed. The coal is sampled to determine its size and analysis and then weighed and placed in the charging bin. The combustion chamber is heated up to about 1700°F by burning a mixture of air and gas, and when the 1700°F temperature is reached, recirculation of products of combustion is begun and the furnace temperature raised to about 2000°F. The coal feed is then started, and the desired rates of feeding and heat supply are adjusted. The system soon comes into balance, which is maintained for the period of the test. The length of a test ranges from 1 to 5 hr, depending upon the rate of feed and object of the test.

During the balanced period, routine operating data are recorded periodically. The temperatures in the combustion system and drying column are recorded continuously, and the pressures in various places, P-1 to P-5 of fig. 3, are determined by direct reading with a portable, water-filled U-gauge. The dried product is removed at regular intervals to determine the rate of yield and to test for moisture removed. When the coal feed stops, the gas circulation is shut off, and all dried coal recovered is weighed and sampled for a material balance.

Before a test is started, theoretical heat and material balances are calculated to establish the desired rate of coal feed, the temperature and velocity of gases in the column, and rates of air, gas, and POC (products of combustion). This calculation is shown in table I, which gives typical calculations for two plants and two coals.

The approximate quantities of air, gas, and coal, calculated as shown, are then fed to the plant. The unit soon comes into operating balance, and the established rates of feed are not changed. Figs. 4 and 5 show the history of temperatures during typical balanced periods when operating plant No. 5.

Results of Tests: Approximately 40 tests have been made on the four fluidized drying units to study various problems of drying different grades of fine coal. Table II presents results of 14 tests showing typical experimental and operating data for seven different coals in plants 3, 4, and 5.

The static pressures in the combustion chamber and the pressure drop across the drying column are shown in table III. The chemical and physical properties of the coals before and after drying are shown in table IV.

Discussion of Results: Experience from operation of the three pilot plants on several grades and ranks of coal has shown that the fluidized drying process is relatively simple. The operation of the units coincides quite closely with theoretical operation as indicated by comparing the calculated data in table I with the experimental data of table II. It is con-

Table II. Summary Data on Drying of Subbituminous Coals and Lignites

Pilot plant	(1)	3	3	3	3	3	4	4
Test number	(2)	8	9-A	10-A	11-A	14	17	18
Rank of coal	(3)	Subc	Subc	Subc	Lignite	Lignite	Subc	Lignite
Source	(4)	Colorado	Alaska	Wyoming	Texas	Greece	Colorado	Texas
Materials and moisture data								
Coal charging rate, lb per hr = W_1	(5)	345	332	277	273	263	1,682	1,166
Coal charging rate, lb per hr per sq ft	(6)	1,716	1,652	1,378	1,358	1,303	2,143	1,485
Moisture in raw coal, pct (as charged)	(7)	24.8	25.8	32.6	37.6	47.3	27.4	39.2
Moisture in dried coal, pct	(8)	4.4	4.2	4.0	4.6	9.0	4.3	3.4
Dry coal recovered, total lb per hr = W_2	(9)	260	236	194	178	151	1,273	727
Improvement ratio by weight loss = R_w	(10)	1.283	1.397	1.428	1.534	1.735	1.321	1.604
Improvement ratio by moisture determination = R_m	(11)	1.271	1.291	1.424	1.529	1.727	1.318	1.589
Ultimate improvement ratio = U	(12)	1.330	1.348	1.464	1.603	1.890	1.377	1.645
Degree of drying, pct = D	(13)	95.9	97.3	91.3	92.0	89.0	98.1	94.5
Dust loss, pct = L	(14)	0.9	0.5	0.4	0.4	0.5	0.3	0.9
Inert gas used for moving coal, cu ft per lb	(15)	1.6	3.2	2.6	2.7	3.3	1.4	1.8
Heating system data								
Natural gas used, cfh (60-30 dry)	(16)	156	161	176	163	197	780	723
Heat supplied, MBtu per hr, net	(17)	137.3	141.7	154.9	143.4	173.4	668.8	636.2
Air used with gas, cfh (60-30 dry)	(18)	1,536	1,656	1,986	1,632	1,896	7,584	7,406
Heat supplied per lb of raw coal, net Btu	(19)	398	427	539	525	662	396	546
Hot gas used to dry coal, cu ft per lb	(20)	9.4	11.5	14.2	13.9	16.4	9.1	13.7
Products of combustion recirculated, cfh	(21)	974	1,346	1,015	1,242	1,319	4,686	4,361
Analysis of gases leaving column:								
H_2O, pct	(22)	38.6	38.1	36.8	40.1	40.9	41.9	44.0
CO_2, pct	(23)	6.9	6.9	6.1	6.7	6.8	6.4	6.1
O_2, pct	(24)	0.7	1.4	2.5	0.8	0.4	0.8	1.1
N_2, pct	(25)	53.8	56.6	54.6	52.4	51.9	50.9	48.8
Mass gas velocity in column, lb per hr per sq ft	(26)	1,597	1,627	1,097	1,909	2,187	2,010	1,992
Mean superficial velocity in column, ft per sec	(27)	11.2	12.4	14.2	13.5	16.7	14.6	14.2
Contact time of coal, sec[a]	(28)	117	122	153	160	174	42	36
Temperatures in system								
Theoretical flame, °F	(29)	2,410	2,190	2,300	2,240	2,380	2,400	2,350
Combustion chamber, °F, (Point 1)	(30)	2,070	1,915	2,005	1,950	2,060	2,080	2,105
Bottom of column, °F, (Point 2)	(31)	290	285	360	345	430	450	460
Top of column, °F, (Point 3)	(32)	275	270	335	300	335	285	300
Gas outlet from primary separator, °F, (Point 5 or 6[b])	(33)	260	255	280	250	325	240	280
Coal in receiver, °F	(34)	225	225	245	215	170		
Heat balance								
Net heat used, Btu per lb of raw coal charged[c]	(35)	321	346	465	498	620	363	498
Net heat required to dry coal, Btu per lb of raw coal[d]	(36)	298	315	418	459	558	327	464
Drying efficiency, excluding radiation, pct[e]	(37)	92.8	91.0	89.9	92.2	90.0	90.1	93.2
Overall efficiency, pct[f]	(38)	74.9	73.8	74.8	87.4	84.3	82.2	85.0

Pilot plant	(1)	4	4	5	5	5	5	5
Test number	(2)	19	21	22-A	29	30-B	31-A	32-A
Rank of coal	(3)	Subc	Lignite	Subc	Subc	Lignite	Lignite	Lignite
Source	(4)	Wyoming	Greece	Colorado	Colorado	Greece	Greece	Texas
Materials and moisture data								
Coal charging rate, lb per hr = W_1	(5)	1,464	810	349	409	142	196	232
Coal charging rate, lb per hr per sq ft	(6)	1,865	1,032	1,736	2,035	706	975	1,154
Moisture in raw coal, pct (as charged)	(7)	30.2	58.8	24.0	24.8	62.9	46.0	37.6
Moisture in dried coal, pct	(8)	3.2	6.8	3.5	2.2	4.6	7.4	5.2
Dry coal recovered, total lb per hr = W_2	(9)	1,053	354	273	313	54	114	149
Improvement ratio by weight loss = R_w	(10)	1.390	2.288	1.278	1.307	2.630	1.719	1.557
Improvement ratio by moisture determination = R_m	(11)	1.387	2.262	1.270	1.301	2.511	1.715	1.519
Ultimate improvement ratio = U	(12)	1.433	2.427	1.316	1.330	2.632	1.852	1.603
Degree of drying, pct = D	(13)	92.3	94.9	88.5	93.2	97.0	90.6	90.8
Dust loss, pct = L	(14)	0.2	1.2	0.7	0.4	4.7	0.3	2.5
Inert gas used for moving coal, cu ft per lb	(15)	1.5	2.6	none	none	none	none	none
Heating system data								
Natural gas used, cfh (60-30 dry)	(16)	707	749	147	212	146	146	150
Heat supplied, MBtu per hr, net	(17)	622.2	659.1	129.4	186.6	128.5	128.5	132.0
Air used with gas, cfh (60-30 dry)	(18)	7,500	7,420	1,470	2,122	1,449	1,449	1,501
Heat supplied per lb of raw coal, net Btu	(19)	425	814	371	456	783	656	569
Hot gas used to dry coal, cu ft per lb	(20)	9.8	17.4	6.9	8.6	16.5	12.2	10.5
Products of combustion recirculated, cfh	(21)	3,938	3,813	762	1,168	737	783	774
Analysis of gases leaving column:								
H_2O, pct	(22)	43.5	46.7	47.0	43.6	50.5	49.2	47.8
CO_2, pct	(23)	5.9	6.0	5.9	6.2	5.8	5.7	5.8
O_2, pct	(24)	1.5	0.7	0.8	0.9	0.7	0.7	0.8
N_2, pct	(25)	49.1	46.6	46.3	49.3	43.3	44.4	45.6
Mass gas velocity in column, lb per hr per sq ft	(26)	1,917	1,939	1,266	1,791	1,299	1,303	1,308
Mean superficial velocity in column, ft per sec	(27)	13.3	14.9	9.1	13.8	9.8	9.3	9.2
Contact time of coal, sec[a]	(28)		19	144	111	208	231	182
Temperatures in system								
Theoretical flame, °F	(29)	2,400	2,530	2,530	2,480	2,560	2,510	2,530
Combustion chamber, °F (Point 1)	(30)	2,140	2,240	2,120	2,120	2,110	2,125	2,100
Bottom of column, °F (Point 2)	(31)	400	990	320	400	315	260	270
Top of column, °F (Point 3)	(32)	280	410	275	335	285	235	240
Gas outlet from primary separator, °F (Point 5 or 6[b])	(33)	270	310	255	305	250	225	225
Coal in receiver, °F	(34)	230	235	235	305	200	220	225
Heat balance								
Net heat used, Btu per lb of raw coal charged[c]	(35)	380	738	307	367	783	532	456
Net heat required to dry coal, Btu per lb of raw coal[d]	(36)	355	684	281	322	715	494	419
Drying efficiency, excluding radiation, pct[e]	(37)	93.4	92.7	91.5	87.7	91.3	92.8	91.9
Overall efficiency, pct[f]	(38)	83.5	84.0	75.7	70.6	79.0	75.3	73.6

[a] Calculated for plant 3 assuming average bed density of 10 lb per cu ft. Other tests calculated on basis of pressure drop.
[b] Point 5 for pilot plants 3 and 4 and point 6 for pilot plant 5.
[c] Total net heat excluding radiation.
[d] Excludes radiation.
[e] Includes radiation.

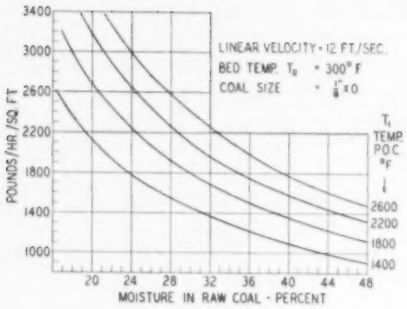

Fig. 6—Capacity of drying column on ¼ in. x 0 coal.

cluded that the probable performance of any unit on any low-rank coals can be calculated in advance.

The data in table II indicate that the maximum practical superficial velocity to achieve 90 to 95 pct moisture removal from ¼ in. x 0 coal is approximately 15 fps. Considerably higher velocities can be attained by reducing the percentage of drying, but for best practical results, superficial velocities between 10 and 15 fps appear to be desirable.

The rapid rate of heat transfer between the hot gases and the fluid bed is shown by comparing temperatures at points 1 and 2, shown in table II. In a distance of less than 6 in., the temperature of the gases drops nearly 1800°F, indicating almost instantaneous evaporation of the moisture from near the surface of the particles. The nearly uniform temperature of the fluid bed is characteristic, but there is a slight decrease in temperature toward the top of the column.

Despite the rapid exchange of heat and the high initial temperature of the hot gases, very little change occurs in the coal substance when removing 90 pct of the moisture. This is illustrated by the analysis of coal before and after drying, as shown on the moisture- and ash-free basis in the data of table V. There is a slight but insignificant reduction in volatile matter averaging about 0.5 pct. The heating value of the coal substance shows very little change, the average heating value increasing from 12,340 to 12,388, an increase of 0.39 pct. The slight devolatilization and increase in heating value indicates some liberation of CO_2, which is one of the first products of thermal decomposition of coal. By increasing the bed temperature to effect 100 pct drying and further devolatilization, the heating value of the treated coal can be substantially increased through liberation of CO_2.

The theoretical performance of a fluidized drying unit, when operating at a superficial linear velocity of 12 fps at sea level, is shown in fig. 6. In this figure, no radiation is considered and 93 pct of the moisture is liberated. The effect of moisture in the coal and the temperature of the initial gases is clearly shown by this graph.

Attention is called to the degradation of the coal during drying. The initial size of the particle and the amount of moisture in the coal influence the extent of degradation. Reference is made to tests 22-A and 29 in table IV on a Colorado subbituminous coal where the degradation is shown to range from 10.6 to 47.6 pct. Relatively little degradation is suffered by the ¼ in. x 0 coal used in test 22-A, but the ½ in. x 0 coal used in test 29 degraded five times as much. The excessive degradation of the large size is caused by rapid shrinkage of the particle which forms big cracks and reduces the strength. The agitation in the boiling bed causes the weakened pieces to disintegrate.

Table III.
Static Pressures in Drying Pilot Plants—Inches H_2O

Plant No.	Test No.	Point 1, Combustion Chamber	Point 2, Above Screen	Point 3, Top of Drying Column	Drying Column Differential	Riser Differential
3	8		5.5		4.0	0.7
3	9-A		5		3.5	0.7
3	10-A		13		9.5	0.3
3	11-A		15		11.5	0.2
3	14	17	10	9.5	0.5	1.0
4	17	18	11.6	9.0	2.6	1.4
4	18	20	12.0	9.5	2.5	1.5
4	19	20	9.2	8.5	0.7	1.1
4	21	19	11.8	10.8	1.0	2.0
5	22-A	19	13	3	10	0.2
5	29	22	16	7	9	0
5	30-B	14	11	5	6	0
5	31-A	17	13	4	9	0
5	32-A	19	15	7	8	1

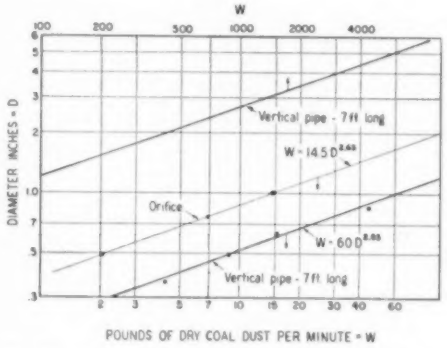
W
100 200 500 1000 2000 4000
6 5 4 3 2 1.0 .7 .5 .3
DIAMETER INCHES = D
Vertical pipe - 7ft long
W = 14.5 D^2.63
Orifice
W = 6.0 D^2.63
Vertical pipe - 7ft long
2 3 5 7 10 15 20 30 40 60
POUNDS OF DRY COAL DUST PER MINUTE = W

Fig. 7—Flow of ¼ in. x 0 dry coal dust by gravity through sharp edged orifices and vertical pipes.

The extreme degradation of Ptolemais lignite shown in test 30-B is interesting. This lignite has an earthy structure, with 62 pct natural bed moisture. The particles apparently explode when heated rapidly. Similar degradation is suffered by the Megara lignite and the Texas lignite. It is probable that, because of the rapid degradation, these low-rank coals might respond favorably in a fluid bed with initial sizes of 1 in. x 0 or more.

The available heat in the several coals before and after drying is shown in table IV. The utilization factor, which is the ratio between the available heat and the gross Btu, is also given. The raw coals have utilization factors ranging from 0.65 to 0.81. This means that 65 to 81 pct of the gross heat in the raw coal can be recovered in an efficient combustion system. However, the dried coals have higher utilization factors, ranging from 0.83 to 0.85, which means that uniform high efficiencies can be obtained from the treated coal.

The physical properties of the dried dusts are similar, which is encouraging for utilization because uniformity is an essential factor in efficient plant performance. It is indicated that a boiler plant designed for burning dried low-rank coals would have

Table IV. Chemical and Physical Properties of Coals Before and After Drying

Drying plant number	(1)	3		3		3		3		3		4		4	
Test Number	(2)	8		9-A		10-A		11-A		14		17		18	
Mine	(3)	Pike View		Suntrana		Wyodak		Sandow		Megara		Sterling		Sandow	
Location	(4)	Colorado		Alaska		Texas		Texas		Greece		Colorado		Texas	
Kind of coal	(5)	Subc		Subc		Subc		Lignite		Lignite		Subc		Lignite	
Condition[a]	(6)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)
Proximate analysis, pct															
Moisture	(7)	24.6	4.4	25.8	4.2	32.6	4.0	37.6	4.6	47.3	9.0	27.4	4.3	39.2	3.4
Volatile matter	(8)	31.9	40.5	37.1	47.0	30.0	42.3	27.2	41.5	22.8	39.2	28.9	39.7	26.5	40.9
Fixed carbon	(9)	37.7	48.0	32.3	42.9	32.1	46.2	27.1	41.6	15.4	26.3	39.4	50.2	23.7	39.3
Ash	(10)	5.6	7.1	4.8	5.9	5.3	7.5	8.1	12.3	14.5	25.5	4.3	5.8	10.6	16.4
Ultimate analysis, pct															
Hydrogen	(11)	6.1	5.0	6.6	5.3	7.0	5.2	7.2	5.0	7.1	4.3	6.5	5.1	7.0	4.8
Carbon	(12)	50.4	64.2	48.6	63.0	46.1	65.2	40.2	61.6	25.3	45.2	52.6	69.3	36.5	58.5
Nitrogen	(13)	0.6	0.8	0.6	0.7	0.7	1.4	0.8	1.2	0.5	0.8	1.2	1.6	0.7	1.2
Oxygen	(14)	37.0	22.5	39.3	24.8	40.4	20.1	42.6	18.3	50.0	21.8	35.1	17.7	43.9	17.3
Sulphur	(15)	0.3	0.4	0.1	0.3	0.5	0.6	1.1	1.6	2.6	4.4	0.3	0.5	1.3	1.8
Btu per lb (gross)	(16)	8,470	10,840	8,450	10,900	7,980	11,330	7,000	10,740	4,250	7,590	9,050	11,970	6,380	10,200
Btu per lb (net)	(17)	7,910	10,380	7,850	10,420	7,340	10,860	6,340	10,280	3,600	7,200	8,460	11,510	5,740	9,760
Ash softening temp., °F	(18)	2,180		2,310		2,130		2,130		2,130		2,150			
Available heat, Btu per lb[b]	(19)	6,880	9,100	6,830	9,160	6,350	9,540	5,440	9,020	2,980	6,260	7,360	10,080	4,920	8,560
Utilization factor[c]	(20)	0.812	0.839	0.808	0.840	0.796	0.842	0.777	0.840	0.704	0.830	0.813	0.842	0.771	0.839
Size consist, cum. pct ret.															
On 0.530 in.	(21)														
Mesh On 0.375 in.	(22)														
(3) On 0.263 in.	(23)														
(4) On 0.185 in.	(24)														
(8) On 0.093 in.	(25)	0.1						0.2		1.5		1.7	0.5	1.9	
(16) On 0.0469 in.	(26)	20.0	7.9	21.4	13.5	26.9	6.1	31.2	2.9	18.4	2.7	22.0	12.1	26.3	5.1
(30) On 0.0232 in.	(27)	52.4	40.7	62.4	51.1	60.5	29.1	65.5	24.8	40.5	27.4	54.0	39.7	59.9	30.0
(50) On 0.0117 in.	(28)	76.3	73.6	83.7	78.6	81.1	56.6	82.4	56.3	60.2	59.1	77.7	67.8	82.7	61.0
(100) On 0.0059 in.	(29)	89.5	89.8	92.9	92.1	90.6	76.5	90.3	75.6	73.2	75.8	88.9	84.4	92.6	77.5
(200) On 0.0029 in.	(30)	96.2	97.4	97.5	97.5	97.3	90.2	96.0	88.8	85.1	86.9	95.1	94.5	99.2	88.7
Avg size, in.	(31)	0.031	0.025	0.034	0.029	0.035	0.020	0.038	0.018	0.027	0.018	0.033	0.025	0.037	0.020
Degradation, pct	(32)		19.4		14.7		42.9		52.6		33.3		24.2		45.9
Bulk density, lb per cu ft															
Modified ASTM	(33)	43.8	44.9	43.2	41.4	40.9	39.6	43.2	39.6	40.6	41.4	45.7	43.9	43.9	42.3
Compacted	(34)	49.8	51.1	51.7	49.6	49.1	45.6	49.2	45.8	54.1	52.7	51.6	50.6	55.4	50.9

Drying plant number	(1)	4		4		5		5		5		5		5	
Test number	(2)	19		21		22-A		29		30-B		31-A		32-A	
Mine	(3)	Wyodak		Ptolemais		Pike View		Pike View		Ptolemais		Megara		Sandow	
Location	(4)	Wyoming		Greece		Colorado		Colorado		Greece		Greece		Texas	
Kind of coal	(5)	Subc		Lignite		Subc		Subc		Lignite		Lignite		Lignite	
Condition[a]	(6)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)
Proximate analysis, pct															
Moisture	(7)	30.2	3.2	58.8	6.8	34.0	3.5	24.8	2.2	62.0	4.6	46.0	7.4	37.6	5.2
Volatile matter	(8)	31.5	42.2	19.4	43.1	31.1	40.9	32.0	41.6	18.1	44.4	23.5	40.1	27.2	40.8
Fixed carbon	(9)	28.9	41.7	14.9	33.3	30.3	48.4	37.4	49.3	13.5	35.4	16.8	28.8	24.4	37.8
Ash	(10)	9.4	12.9	6.9	16.8	5.6	7.2	5.8	6.9	6.4	15.6	13.7	23.7	10.8	16.2
Ultimate analysis, pct															
Hydrogen	(11)	6.6	5.0	8.3	4.8		4.9		4.6		4.5		4.3		4.7
Carbon	(12)	45.1	61.4	23.0	50.8		64.8		66.1		53.3		45.3		57.1
Nitrogen	(13)	0.6	0.9	0.6	1.5		0.8		0.8		1.5		0.8		1.1
Oxygen	(14)	37.5	18.6	60.8	24.9		21.9		21.2		23.8		21.5		18.8
Sulphur	(15)	0.8	1.2	0.7	1.2		0.4		0.4		1.3		4.4		2.1
Btu per lb (gross)	(16)	7,740	10,760	3,820	8,640		10,940		11,290		8,870		7,680		9,980
Btu per lb (net)	(17)	7,140	10,300	3,060	8,200		10,490		10,870		8,460		7,290		9,550
Ash softening temp., °F	(18)														
Available heat, Btu per lb[b]	(19)	6,170	9,050	2,490	7,170	7,210	9,210	6,850	9,580	2,160	7,400	3,210	6,350	5,060	8,380
Utilization factor[c]	(20)	0.797	0.841	0.652	0.830	0.816	0.842	0.811	0.848	0.624	0.834	0.715	0.827	0.774	0.840
Size consist, cum. pct ret.															
On 0.530 in.	(21)							3.6		1.1		1.8			
Mesh On 0.375 in.	(22)							10.0	1.5	4.6		4.7		1.1	
(3) On 0.263 in.	(23)							21.0	3.0	12.8		13.9	0.3	9.8	0.4
(4) On 0.185 in.	(24)							31.0	7.3	27.0		29.1	0.6	25.8	1.9
(8) On 0.093 in.	(25)	1.9	0.3	4.6		1.2	0.9	58.2	33.5	53.9	0.5	58.1	4.6	59.4	14.3
(16) On 0.0469 in.	(26)	37.6	14.5	29.4	2.3	16.6	11.5	77.9	62.9	71.6	6.9	78.0	34.8	77.6	39.5
(30) On 0.0232 in.	(27)	70.4	46.4	54.5	24.4	46.0	41.3	89.4	81.5	82.7	44.2	85.0	61.1	87.7	66.4
(50) On 0.0117 in.	(28)	86.9	73.8	73.5	54.0	70.5	68.5	95.5	91.8	90.9	71.0	91.7	81.6	93.7	83.3
(100) On 0.0059 in.	(29)	93.1	87.2	85.9	71.6	86.2	85.4	98.0	96.5	95.9	84.4	95.4	90.0	96.9	91.2
(200) On 0.0029 in.	(30)	96.4	94.7	98.9	86.9	96.0	98.0	99.5	99.1	99.6	93.7	97.9	94.9	98.7	96.2
Avg size, in.	(31)	0.043	0.028	0.038	0.017	0.057	0.051	0.166	0.087	0.137	0.035	0.147	0.039	0.133	0.053
Degradation, pct	(32)		34.9		55.3		10.6		47.6		51.8		73.5		60.2
Bulk density, lb per cu ft															
Modified ASTM	(33)	44.7	42.7	35.4	38.7	45.5	44.7		42.7	37.7	35.0	44.1	42.8	43.0	40.8
Compacted	(34)	53.0	50.5	48.8	46.7	52.8	51.9		50.0	51.4	43.6	51.9	52.7	50.7	47.2

[a] (1) Before drying; (2) After drying.

[b] Burned with 30 pct excess air, and stack temperature = 500°F.

[c] Ratio of available heat to gross Btu per lb.

Table V. The Effect of Drying as Shown by the MAF Analysis of the Raw and Dried Coals

Test No.	Mine County and State	Condition[a]	Proximate Analysis		Ultimate Analysis					Btu per Lb	Drying Temp., °F.[b]
			Volatile Matter	Fixed Carbon	Hydrogen	Carbon	Nitrogen	Oxygen	Sulphur		
8	Pike View El Paso County, Colo.	(1) (2)	45.9 45.7	54.1 54.3	4.9 5.1	72.5 73.5	0.9 1.0	21.3 21.0	0.4 0.4	12,170 12,240	275
9-A	Suntrana Suntrana, Alaska	(1) (2)	53.3 52.3	46.7 47.7	5.4 5.5	70.1 70.0	0.8 0.8	23.6 23.3	0.1 0.4	12,160 12,130	270
10-A	Wyodak Campbell County, Wyo.	(1) (2)	48.2 47.9	51.8 52.1	5.3 5.4	74.3 73.6	1.0 1.6	18.6 18.7	0.8 0.7	12,850 12,800	335
11-A	Sandow Milan County, Tex.	(1) (2)	50.1 49.9	49.9 50.1	5.5 5.4	74.1 74.1	1.4 1.4	17.0 17.2	2.0 1.9	12,900 12,910	300
14	Kokkinarea Near Megara, Greece	(1) (2)	59.7 57.9	40.3 42.1	4.9 4.9	66.3 66.9	1.3 1.2	20.8 20.6	6.7 6.4	11,140 11,250	315
17	Sterling Weld County, Colo.	(1) (2)	42.1 44.1	57.9 55.9	5.0 5.1	76.9 77.2	1.7 1.7	15.9 15.5	0.5 0.5	13,250 13,310	285
18	Sandow Milan County, Tex.	(1) (2)	52.8 51.0	47.2 49.0	5.4 5.4	72.6 72.9	1.4 1.4	18.0 18.1	2.6 2.2	12,680 12,710	300
19	Wyodak Campbell County, Wyo.	(1) (2)	52.1 50.3	47.9 49.7	5.4 5.6	74.5 73.2	1.0 1.0	17.7 18.7	1.4 1.5	12,800 12,840	280
21	Ptolemais area, Greece	(1) (2)	56.7 56.4	43.3 43.6	5.0 5.3	66.9 66.5	1.9 2.0	24.2 24.6	2.0 1.6	11,110 11,300	410

[a] Condition: (1) Before drying. (2) After drying.
[b] At top of column.

uniform operating characteristics despite the source of raw coal.

The dried dusts are free flowing, and since the moisture content is lower than the equilibrium value, they are hygroscopic and their tendency to pack in a bin is reduced provided moisture is excluded. Interesting experiments were made on the movement of dried dusts through orifices and vertical pipes. These are illustrated in fig. 7. It is shown that the dried dusts move through orifices similar to liquids, and a formula is derived showing relationship between size of opening and rate of discharge. The weight of dried dust that will discharge through an orifice per minute is equal to $14.5\ D^{2.65}$, D being the diameter of the orifice in inches. Similar experiments were made to find out how fast dried dust will move through vertical pipes. Pipes of various lengths and diameters were connected to a bin and the discharge measured. It was found that a pipe 7-ft long had the greatest capacity, and the amount of dust, in pound per minute, that would flow through a 7-ft vertical pipe from a bin is equal to $60\ D^{2.65}$. In other words, a vertical pipe 7-ft long will discharge about four times as much dried dust as an orifice of equal diameter. This phenomenon is due to the energy developed by the falling particles, which entrains gas and acts as a siphon. The phenomenon may be employed for efficient handling of dried dusts in large plants.

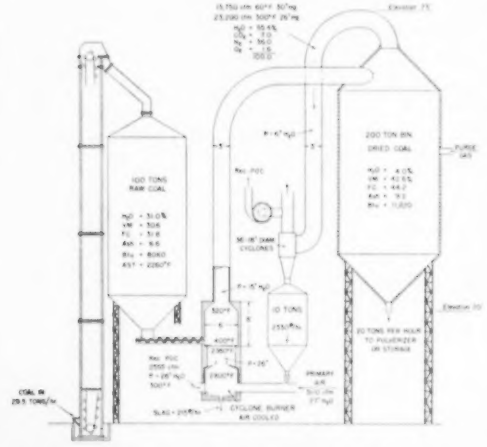

Fig. 8—Schematic design of drying unit for subbituminous coal. Capacity 20 tons of dried coal per hour.

Commercial Application of the Fluidized Drying Process

Experimental work on several pilot plants has carried the investigation about as far as feasible on a small scale. The next step is to operate a commercial-size unit to determine engineering and economic data. Fig. 8 shows a plant designed to dry 30 tons of Wyoming subbituminous coal per hour and to deliver about 20 tons of dried coal per hour. This schematic arrangement shows the essential ingredients of a plant and indicates the flowsheet and quality of materials.

The cyclone burner for generating primary hot gases under pressure may or may not be applicable for such service, and some development work would be necessary to work out a suitable burner for this drying unit. The problem of burning pulverized coal under pressure is not difficult, and existing equipment can be readily adapted to solve this problem.

The unit illustrated in fig. 8 is estimated to cost approximately $100,000. The investment and operating costs of three units of this design for processing 2000 tons of coal per day, when removing 670 lb of water per ton, are estimated as follows, in cents per ton: investment cost, 7.0; labor and supervision, 6.2; power, 4.5; fuel, 9.8; maintenance, 1.1; total, 28.6.

These estimated costs, which do not include the cost of crushing the coal, indicate that moisture can be removed from lignite at a cost of about $0.85 per ton of water removed. The cost will vary with the moisture content of the coal treated, but it is believed that the unit cost for large commercial plants will be in the range of $0.85 to $1.00 per ton of moisture removed.

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aime NEWS

Industrial Minerals Division to Hold Three Big Regional Meetings in October

BEGINNING on Oct. 12, and continuing through Oct. 20, the IMD will hold three regional meetings which will bracket the western United States. At Norman, Okla., El Paso, Texas, and in Los Angeles, the IMD will be ready with good technical programs and fine social activities.

The Norman Meeting

This regional gathering will be held Oct. 17-20 at the University of Oklahoma Extension Study Center, North Campus, on the north edge of Norman, just 20 miles from Oklahoma City. At least sixteen papers have been scheduled thus far. Plans call for an active discussion of the titanium papers by men familiar with the occurrence of ilmenite deposits. **The meeting program appears on p. 984.**

Field trips will be made to the Arbuckle Mountains, south-central Oklahoma, on Thursday and Friday, Oct. 19 and 20. The first day will include inspection of industrial minerals deposits and operating plants, including glass sand, dolomite, cement materials, asphaltic rock and crushed stone. The second day will be devoted to stratigraphic and structural features in lower Paleozoic sedimentary rocks of the central and Western Arbuckle Mountains. An alternate trip to the Wichita Mountains, to study pre-Cambrian basic rocks and titaniferous iron ores, is a possibility if there are sufficient registrants. It is important that those who wish to make the field trips inform Robert H. Dott, Oklahoma Geological Survey, Norman, Okla., as soon as possible.

No housing problems are anticipated at Norman, as there are 34 double rooms at the Study Center, and 62 rooms available in adjoining buildings. Rates are $1.50 to $2 per person per night. In Norman, the Norman Court Hotel will have rooms available at rates of $3.50 single and $2 double. Make your reservations with John B. Freeman, Extension Division, University of Oklahoma, Norman.

A banquet will be held Tuesday night, Oct. 17, in the dining room at the Study Center—at the very reasonable rate of $1.50.

Rocky Mountain Group Meets at El Paso

The IMD's Rocky Mountain regional meeting will hold forth in El Paso over the Oct. 12-14 week-end, in conjunction with the International Mining Days celebration in that city. A series of six papers will be presented. The El Paso Metals Section, AIME, will also be represented at the three-day celebration, and will hold their Annual Banquet on Thursday night, Oct. 12. Over 350 people attended the banquet last year, and those who come in October will again be regaled by the top-notch humorist, Jeff Williams, who did such a fine job last year. The Rancheros breakfast is on the agenda for Friday morning.

A forty-six man committee is at work in El Paso preparing the Mining Days celebration. At the top of their list is the Suppliers Party to be held on Friday night. For the sports lovers, there's the University of Arizona-Texas Western football game, to be played under the lights on Saturday night.

Southern California

The Industrial Minerals Division will join all divisions and branches of the Southern California Section in Los Angeles Oct. 12-13. Although no final program has been set as we go to press, it is understood that a paper on the Kramer operation of the Pacific Coast Borax Co. is scheduled, thanks to some fine work by IMD Program Chairman Conrad Thomas. State Geologist Olaf P. Jenkins has agreed to present a paper on commodity studies in industrial minerals. A contribution on iodine production, and one on methods of brine extraction from Searles Lake, will also be on the program.

Programs

Rocky Mountain Regional Meeting
Industrial Minerals Division, AIME

In conjunction with the International Mining Days Celebration, El Paso, Texas, October 12-14

Sinking a Shaft Through Pre-Grouted Ground. G. T. Harley
Transition in Recent Developments in Nonmetallic Minerals. W. E. Gay
Fluorspar in Arizona. E. D. Wilson
Some Features of Perlite in New Mexico. E. Callaghan
Clay in New Mexico. Staff member, U.S.B.M.

Industrial Minerals Division Fall Regional Meeting

Norman, Oklahoma, Oct. 17-20

Technical Papers

Trends in the Industrial Mineral Development of the Southwest. Harold N. Foxhall

Geology and Industrial Minerals of Oklahoma. Robert H. Dott

Chemical Grade Limestone and Dolomite in Oklahoma. Homer Dunlap

Industrial Minerals of the Arbuckle Mountains, Oklahoma. William E. Ham

Titaniferous Magnetite in the Basic Rocks of Wichita Mountains, Okla. Gerald W. Chase

Geology and Utilization of the Titaniferous Magnetites at McIntyre Development, Tahawus, New York. George W. Wunder and Paul W. Allen

Chats in the Tri-State District. Ernest Blessing

Salt in Kansas. Earl Nixon

Specifications for Specific Uses of Limestones and Dolomites. Edward Clark and Mabel E. Philips

Geology and Ceramic Possibilities of Wilcox Clays of Central Arkansas. Norman F. Williams

Barite Production in Missouri. Garrett A. Muilenberg

Drama of Portland Cement. Cecil Perkins

Industrial Minerals of Texas. John T. Lonsdale

Gulf Coast Sulphur an Industrial Mineral. Albert G. Wolf

Production of Anhydrous Sodium Sulphate from Natural Brines. William I. Weisman and R. C. Anderson

Geologic Setting for Potash Deposits in Southeast New Mexico. Joe P. Smith

Dinner Speaker

Dr. Hugh D. Meiser, USGS. **Making a Geologic Map of Oklahoma**

ANNUAL MEETING COMMITTEEMEN CHOSEN

Time: February 19-22, 1951

Place: St. Louis, Mo.

Headquarters: Hotel Jefferson

Meetings: Hotel Jefferson and Hotel Statler

The AIME Annual Meeting will be held in St. Louis, Mo. Feb. 19 to 22, 1951, under the General Chairmanship of Howard I. Young. Mr. Young is a director of AIME, and president of the American Zinc, Lead & Smelting Co. Secretary for the meeting is Fred J. Meek, plant engineer of the American Zinc Co. of Ill.

The committees have been chosen, and work is underway to complete all necessary details.

The major part of the planning will be done by members of the St. Louis Section. Papers for presentation will be selected by the Technical Publications Committee. The program schedule will be arranged by the Coordinating Committee on Technical Program for the Annual Meeting, under the Chairmanship of S. J. Swainson. Present plans call for a student forum under the guidance of faculty sponsors of student chapters. Because of the large program, some technical meetings will be held at the Hotel Jefferson, and some, including those of the Metals Branch, will be held at the Hotel Statler.

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James D. Forrester
O. R. Grawe
A. P. Green
W. T. Isbell
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AIME-ASME Fuels Conference Starts Oct. 24 in Cleveland

The Coal Division, AIME, will join forces with the American Society of Mechanical Engineers at the Statler Hotel in Cleveland Oct. 24-25 for their Thirteenth Annual Fuels Conference. Technical sessions will feature more than thirteen papers, and two luncheons and a banquet will fill out the two-day program. The banquet is set for Tuesday night, Oct. 24, at the Statler, and will be preceded by a cocktail hour.

Top-notch luncheon and banquet speakers have been procured, with Cleveland's commissioner of air pollution, H. G. Dyktor, outlining the work of smoke inspectors, Elmer Lindseth, president of the Cleveland Electric Illuminating Co., speaking on "The Kilowatt Hour in War and Peace", and George Whitwell, vice-president of the Philadelphia Electric Co., addressing Wednesday's luncheon meeting. At the banquet, the Percy Nicholls award will be presented to J. E. Tobey, and a 50-year membership award will be presented by the ASME to John Rowland Brown.

Plans for ladies' events have been made by the Ladies Auxiliary Committee, ASME, Cleveland Section, and include a tea at the Statler and a luncheon and entertainment at the Carter Hotel.

Program

Thirteenth Annual Fuels Conference, AIME-ASME

Fuels Division, Cleveland Engineering Society, Cooperating

Statler Hotel, Cleveland, Ohio, Oct. 24-25

October 24

Registration, 9 AM

10 AM—SESSION I, EUCLID BALLROOM

Panel—Coal Characteristics Affecting Burning Equipment

Underfeed Stokers. Representative of American Engineering Co.

Pulverized Coal. Roger Curfman

Spreader Stokers. Charles Hallman

Chain Grate Stokers. Harry Huston

Cyclone Furnaces. Francis X. Gilg

Gas Analyzers for Better Combustion. C. H. Barnard

12:30 PM—LUNCHEON I, GRAND BALLROOM

Presiding: Elmer R. Kaiser, Chairman, Fuels Div., ASME

Speaker: H. G. Dyktor, Commissioner, Div. of Air Pollution, Cleveland

Subject: Meet the Smoke Inspector

2 PM—SESSION II, EUCLID BALLROOM

Carl Mabley and A. W. Thorson, Co-Chairmen

The Causes and Probable Effects of Today's Fuel Situation. H. A. Baldwin

A Pattern for Sound Utility Fuel Procurement. Marshall Pease, Jr. and R. J. Brandon

6 PM—COCKTAIL HOUR

7 PM—BANQUET, GRAND BALLROOM

Toastmaster: Thomas A. Marsh, Iron Fireman Mfg. Co.

Speaker: Elmer L. Lindseth, president, Cleveland Electric Illuminating Co.

Subject: The Kilowatt Hour in War and Peace

AWARDS: Percy Nicholls Award to J. E. Tobet, president, Appalachian Coals, Inc.

50-year member award to John Rowland Brown, president, Reliance Gage Column Co.

October 25

10 AM—SESSION III, EUCLID BALLROOM

Ralph Textor and George D. Keller, Co-Chairmen

Coal Sampling by Large Increment Weights. B. A. Landry and W. W. Anderson

The Effect of Coal Size on the Sampling of Coal for Float and Sink Tests. A. L. Bailey and B. A. Landry

12:30 PM—LUNCHEON II, GRAND BALLROOM

Presiding: Carroll F. Hardy, Secretary, Fuels Div., ASME

Speaker: George E. Whitwell, vice-president, Philadelphia Electric Co.

2 PM—SESSION IV, EUCLID BALLROOM

James Hyslop and Richard H. Swallow, Co-Chairmen

Frothing Characteristics of Pine Oils in Flotation. Shiou-Chuan Sun

Dewatering of Fine Coal. C. C. Wright

An Investigation of the Abrasiveness of Coal and Its Associated Impurities. H. F. Yancey, M. R. Geer, and J. D. Price

Ladies' Events

Mrs. Wm. Stewart, Chairman. Mrs. W. G. Waltermire, Vice-chairman. Mrs. Robert L. Manley

October 24—Tea, Statler Hotel, 3-4:30 PM

October 25—Luncheon and entertainment, Carter Hotel

Women's Auxiliary Dinner Meeting

Dr. Howard A. Rusk, associate editor of the New York Times and professor and chairman of the department of physical medicine and rehabilitation, New York University-Bellevue Medical Center, will be the speaker October 25 at a dinner meeting of the New York Section of the Women's Auxiliary of the AIME at the Engineering Women's Club, 2 Fifth Ave. Speaking on the subject, "Untapped Manpower; The Handicapped in Mobilization," Dr. Rusk will discuss the rehabilitation programs at the New York University-Bellevue Medical Center, which the Women's Auxiliary have assisted since their organization.

Reservations may be made by sending check for $3.25 to Mrs. Clifford Bowden, Jr., 2104 Albemarle Terrace, Brooklyn 26.

Drift of Things . . . . . as followed by *Edward H. Robie*

Start of A Korean War

"My first intimation of trouble came early on Sunday afternoon, June 25, when I was called to the Embassy and told that the North Korean Army had crossed the border in force, 30 miles away. While this was the largest scale operation yet reported on the border, we were told that the South Korean forces would undoubtedly shoo them back to the north within a matter of hours, as they had done repeatedly before.

"The evening news was encouraging and our military observers at the front reported that the enemy reconnaissance forces had been stopped. During the night, however, enemy infiltrations succeeded in cutting communications between Korean Army units,

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A FIRST-HAND account, from an AIME member, of the first few days of the Korean War as experienced by a resident of Seoul was received late in July. The writer, Don L. Carroll, was for a time associate editor of Oil Weekly, and recently has been with the Oil and Gas Div., of the Dept. of the Interior. He was a member of the ECA mission to Korea. His wife, Catharine, and children were with him in Seoul when the war started.

---

and confusion began to develop. By 2:30 a.m. on Monday, the enemy forces had penetrated to within 17 miles of Seoul, and the Ambassador issued the order to evacuate wives and children to Ascom, about halfway to the port of Inchon. This assured easier evacuation from Seoul itself in case of eventual full evacuation of all Americans in Korea. By 6 p.m. of the same day, all dependents had been placed aboard the Norwegian freighter Reinhold in Inchon harbor. C rations, blankets, water, etc., were loaded aboard and the ship sailed for Japan an hour later. Fighter planes had been sent over from Japan to cover the whole operation and to escort the ship at sea.

"This protection was necessary because Communist planes had been strafing Seoul, Inchon, and the airports at nearby Kimpo and Yongdongpo most of the day. One Red plane crossed Seoul after I had seen my brood safely aboard the ship and had returned home to do some hasty packing for myself. The plane had just shot up the Yongdongpo airfield about 3 miles from our house and had then come in over town at a low level with guns blazing away. I had run out into the backyard to watch for a moment, but when I noted that our house was in the line of fire and observed tiles flying off the roof, I immediately returned to my packing, and with some enthusiasm. I finished with my two bags (no more allowed) and prepared for a potential night's rest. I had a good start towards that objective when I was called back to the Embassy for around-the-clock duty. I could hear the battle noise north of town as I drove downtown through the blackout.

"By 6:30 p.m. on Tuesday this operation was essentially completed and I had destroyed my files except for the contents of my briefcases, which represented all that I know, and which I therefore felt I must salvage. Who wants to be uninformed when he is supposed to be an expert? At 7 a.m. it was decided to evacuate nonessential mission personnel, including all women employees who had been kept on the job. Transport planes and fighter escort had been ordered from Japan to evacuate this group from Kimpo airport. In spite of air attacks at Kimpo during this operation, all of this group were safely evacuated to Japan by noontime. Three of Uncle Joe's angels descended from the heavens to interrupt this operation, and continued right on down into the ground as a result of the active interest of our own F-80 fighters.

"As the above group were being assembled at the Embassy, I was told that because of my family status I was eligible to be included, and that I could leave with the group 'if I insisted.' So, I stayed on the job. By 9 a.m. Tuesday, the Reds had moved up to the outskirts of Seoul, and with tanks. Since the South Koreans had run short of ammunition, had no anti-tank guns, and no effective staff control, we knew that we must now take it on the lam, as it were. More transport planes were requested from Japan, and at 10:15 we took to the busses to rendezvous with the planes at Kimpo. During our last hour at Seoul, I joined a team swinging sledgehammers at the Embassy. We wrecked the radio station, telephone exchange and message center, and completed destruction of files with incendiary bombs which we set off in the piles of papers in the parking lot.

"I remember thinking at that point that I would feel much more like being evacuated if I could only have a shower. I was practically out on my feet, since as my children insist, I am an old man now. Even so, I had the situation right in the hollow of my head, just up to the level of the hole that penetrates it.

"I did get the shower, though. A few minutes later after our motor convoy got underway headed for Kimpo. It was a shower of bullets from a Yak fighter that buzzed up right on the main drag in town. No one was hit in our convoy, but Koreans on the other side of the street were mowed down like grass. We had expected to be attacked as we were making the slow crossing of the long Han River bridge at the south edge of town since it had been jammed with refugees for hours, all heading south. It required 20 min to negotiate the crossing, after which we began to breathe abnormally again.

"By 1:30 p.m. our own fighters appeared over Kimpo and set to work shooting down or chasing away all enemy aircraft within ten miles of the spot. The transport planes came in promptly after the way had been cleared and by 4 p.m. we were all on our way to Japan, which is a very beautiful country at the end of June.

"I have been told that Tiff distinguished himself on the ship, getting women and their luggage aboard and carrying small children up the ladder. He spent the night organizing and serving with the young men in white—a night bucket brigade, which was necessary because of the fact that there were only two single latrines on the ship, for some 700 women and children. Tiff and three of his strong-stomached pals carried the buckets from the live cargo hold to the rail and return, ad nauseum and ad infinitum. I was very proud of him. He wasn't spoiled after all—he just smelled that way.

"Maxie naturally spent his waking hours in the radio room, and enjoyed the whole voyage. Julie helped entertain the 'little' children. Catharine just held back on her respiration and kept her mouth closed—a difficult feat for her, and one that distinguished her from the run of women aboard, many of whom pulled out all the stops, pulled the rank of their husbands, and complained to the Captain about their accommodations. The gamut was just as long as regards the actions and reactions of the menfolk. I was amazed at some of them, and increased my respect for others, although few of the shots could have been called in advance. I could write a book about it.

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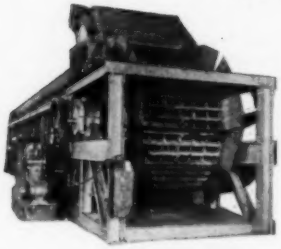

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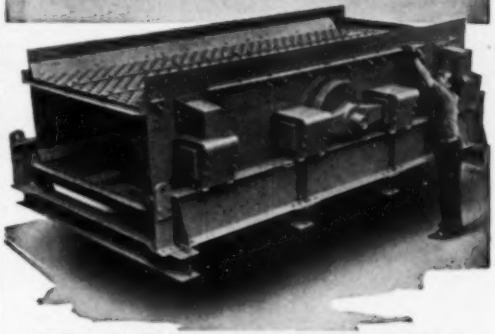

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Coming Events

Sept. 1, AIME, Minerals Beneficiation Division, Hotel Utah, Salt Lake City.

Sept. 3-8, American Chemical Society, National Chemical Exposition, Chicago Coliseum, Chicago.

Sept. 5-9, American Chemical Society, 6th national chemical exposition, Chicago Coliseum, Chicago.

Sept. 10-13, American Institute of Chemical Engineers, Radisson House, Minneapolis.

Sept. 12, Society for Applied Spectroscopy, Socony-Vacuum Training Center, New York.

Sept. 15-16, National Society of Professional Engineers, Wheeling, W. Va.

Sept. 18-22, Industrial Safety Institute, Pennsylvania State College, State College, Pa.

Sept. 19-21, American Society of Mechanical Engineers, fall meeting, Hotel Sheraton, Worcester, Mass.

Sept. 20, AIME, Carlsbad Potash Section, Riverside Country Club, Carlsbad, N. Mex.

Sept. 22-23, AIME, El Paso Metals Section, Chihuahua, Chih., Mexico.

Sept. 25-26, Ohio Mineral Industries Conference, Ohio Museum Auditorium, Columbus.

Sept. 28, AIME, Lehigh Valley Section, visit to Lehigh Navigation Coal Co., Lansford, Pa.

Oct. 2-5, American Institute of Electrical Engineers, Baltimore.

Oct. 5, AIME, Ajo Subsection, Ajo, Ariz.

Oct. 4-6, AIME, Petroleum Branch, Roosevelt Hotel, New Orleans.

Oct. 11-12, AIME, Industrial Minerals Div., Rocky Mt. Section, El Paso, Texas.

Oct. 12, AIME, El Paso Metals Section, El Paso, Texas.

Oct. 12-13, AIME, Southern California Section, Metal, Mining, and Petroleum Branches, Elks Club, Los Angeles.

Oct. 13, AIME, Southwestern Section, Open Hearth Committee, Iron and Steel Div., Houston, Texas.

Oct. 13-14, International Mining Days, El Paso, Texas.

Oct. 16-20, National Safety Congress and Exposition. Industrial safety sessions, Stevens, Congress, and Morrison Hotels; traffic safety, Congress; commercial vehicle, farm and home safety, La Salle; school, Morrison.

Oct. 17-20, AIME, Industrial Minerals Div., fall meeting, Norman, Okla.

Oct. 20, AIME, Eastern Section, Open Hearth Committee, Iron and Steel Div., fall meeting, Warwick Hotel, Philadelphia.

Oct. 20-21, Engineers' Council for Professional Development, annual meeting, Hotel Tudor Arms, Cleveland.

Oct. 22-24, American Mining Congress, metal and nonmetallic convention, Biltmore Hotel, Los Angeles.

Oct. 23-25, AIME, Coal Div., and **ASME,** Fuels Div., Hotel Statler, Cleveland.

Oct. 23-27 National Metal Congress and Exposition, International Amphitheater, Chicago. Participating organizations: AIME, Headquarters, Hotel Sheraton; ASM, Headquarters, Palmer House; American Welding Society, Headquarters, Hotel Sherman; Society for Non-destructive Testing.

Oct. 27, AIME, Southern Ohio Section, Open Hearth Committee, Iron and Steel Div., annual meeting, Deshler-Wallick Hotel, Columbus, Ohio.

Nov. 3, AIME, Pittsburgh Section of Open Hearth Committee, Iron and Steel Div., and Pittsburgh Section, AIME, annual meeting, William Penn Hotel, Pittsburgh.

Nov. 3-5, New Mexico Geological Society, field conference, San Juan Basin.

Nov. 9, American Mining Congress, Coal Div. Conference, William Penn Hotel, Pittsburgh.

Nov. 14, AIME, Buffalo Section, Open Hearth Committee, Iron and Steel, Div., all-day meeting, Statler Hotel, Buffalo.

Nov. 16-18, Geological Society of America, annual meeting, Hotel Statler, Washington, D. C.

Nov. 24, AIME, Central Appalachian Section and W. Va. Coal Mining Assn., annual meeting, Greenbrier, White Sulphur Springs, W. Va.

Nov. 26-Dec. 1, American Society of Mechanical Engineers, annual meeting, Hotel Statler, New York.

Nov. 27-Dec. 2, National Exposition of Power & Mechanical Engineering, Grand Central Palace, New York.

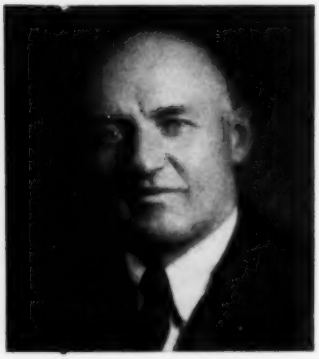
Dec. 7-9, AIME, Electric Furnace Steel Conference, Iron and Steel Div., Hotel William Penn, Pittsburgh.

Jan. 15-17, 1951, AIME, Minnesota Section, annual meeting. Mining symposium conducted by Center for Continuation Study, University of Minnesota.

Feb. 18-22, 1951, AIME, annual meeting, Jefferson Hotel, St. Louis. Metals Branch session to be held at the Statler Hotel.

AIME Personals

David Ailman has joined the staff of the mineral dressing division at the Stamford research laboratories, American Cyanamid Co., Stamford, Conn.



Arthur J. Blair

Arthur J. Blair has retired as chief geologist with Tennessee Coal, Iron & Railroad Co., Birmingham. He has opened his office as a consulting and mining geologist, 620 First National Bldg., Birmingham.

Alex A. Briber shall assume duties as a research fellowship student at the University of Washington in the fall. He will be working in the field of coal preparation.

Gerald G. Brun is now employed as a sales engineer with United Clay Mines Corp., Trenton, N. J.

Jessie C. Carlile is now employed as a field engineer with the Colorado Iron Works, Denver.

Stewart Carpenter left New York on July 27 for Chuquicamata, Chile, where he will be assistant chief engineer in charge of construction for the new sulphide plant, including both a mill and smelter. He has been in Ajo, Ariz. for the last year, building the new Phelps Dodge smelter which went into operation this summer.

John R. Castano is now a geological trainee for the Shell Oil Co., Casper, Wyo.

James D. Crawford has been made manager of the Fairbanks operations of the U. S. Smelting, Refining & Mining Co. He will succeed **R. H. Ogburn** who is retiring after 25 years with the company. **J. C. Boswell** has been appointed assistant manager.

George A. Davidson, mining engineer for the American Cyanamid Co., is leaving for Bihar Province, India where his company is erecting an ammonia-sulphate plant for the government of India. The plant is scheduled to be in operation early in 1951 and Mr. Davidson expects to remain in India about a year.

John H. Dismant, formerly assistant professor of metallurgy, New Mexico School of Mines, Socorro, N. Mex., is now a fellow in metallurgy at the University of Utah, Salt Lake City.

Elmer R. Drevdahl, Jr., is doing graduate work in mining. He has a fellowship doing research at the University of Washington in cooperation with the U. S. Bureau of Mines.

A. W. Edelen is retiring as assistant general manager, mining department, American Smelting & Refining Co., Mexico City. He will continue as a consultant and may be reached at San Juan de Letran No. 11, Mexico, D. F., c/o Cia Minera del Mexcala.

Nevzat Mehmet Erkun, formerly with Pittsburgh Forgings Co., Corapolis, Pa. has returned to his home in Kirikkale, Turkey.

Charles F. Fogarty, connected with Socony-Vacuum, was the recipient of a fellowship award. He will study for his master's degree in geology.

Evar John Forslund has left the employ of the Cerro de Pasco Copper Corp., Cerro de Pasco, Peru.

Walter F. Galloway is now superintendent of metallurgical operations, Polaris-Taku Mining Co., Ltd., Tulsequan, B. C., Canada.

Walter P. Gillingham has been employed as a shoveler by the New Jersey Zinc Co., Gilman, Colo.

Robert F. Griffith is employed as a project engineer with the U. S. Bureau of Mines, SMIB, at Boise.

A. F. Halcrow, exploitation engineer, has been transferred from the Canadian office of Shell Oil to Houston.

Charles V. Harris is now employed as mill superintendent, Blue Star Mines, Ltd., Big Pine, Calif.

James G. Henderson is now divisional superintendent, Fresnillo Mining Co., Fresnillo, Zacatecas, Mexico. He formerly held the position of chief engineer, Minas de Matahambre, S. A., Provincia de Pinar del Rio, Cuba.

George Hawn

George Hawn has been employed by Compania Minera Unificada del Cerro de Potosi, Potosi, Bolivia, as mine superintendent. He was formerly connected with the Newmont Mining Corp., Heber, Utah.

A. D. Hughes of San Francisco, recently returned from a 3½ months trip to French Equatorial Africa. He was investigating alluvial gold and diamond mines for the Bureau Minister de la France d'Outre-Mer.

Edmond A. Krohn is now employed as a foreman with the Robinson Brick & Tile Co., Denver.

Charles H. Lambur, president of United States Collieries, Inc., sailed in June for Europe on various consulting work in mining operations in Turkey, Greece, Italy, and France. He will return in October.

Ralph E. Larry and **Arthur W. Worthington,** vice-president and president respectively, Pittsburgh Limestone Corp., New Castle, Pa., have retired.

A. Lewis is now field engineer for American Smelting & Refining Co., Tonopah, Nev. He is in charge of diamond drill development on a group of claims 1½ miles north of Tonopah.

J. T. Lewis, Jr., succeeds Mr. Duriez as manager of the Bayard department, U. S. Smelting, Refining & Mining Co., Bayard, N. Mex. Mr. Lewis previously held the position as superintendent of the Ophir unit of that company at Ophir, Utah.

Theodore A. Link has formed a partnership with **Arthur W. Nauss**

as exploration consultants, oil and gas appraisers, etc.

Frank Marold has been transferred to Washington, D. C., as a sales representative for Allis-Chalmers Mfg. Co. Prior to his transference he was in the export department in the Milwaukee plant.

J. K. MacFarlane has been appointed superintendent of the Scrub Oaks department, Alan Wood Steel Co., Dover, N. J.

Charles E. Melbye recently graduated from the Colorado School of Mines. He is now employed as a sampler and junior geologist by Anaconda Mining Co., Shoshore mines, Tecopa, Calif.

Donald H. McLaughlin, president of the Homestake Mining Co. and also AIME president, and **J. A. Martino,** president, National Lead Co., were among those who received citations at the Boston Jubilee.

Thomas G. Moran is presently employed by the Comision Ejecutiva Hidroelectrica Del Rio Lempa as geologist and foundations engineer on the Rio Lempa dam project in San Salvador, El Salvador.

Cyril W. Morris

Cyril W. Morris has opened his offices as a consulting mining engineer and may be contacted at 15 Maitland St., Killara, N.S.W., Australia.

Roderick G. Murchison, Jr., has accepted an appointment as petroleum geologist under the U. S. Geological Survey-British Colonial Survey cooperation program. He will be located in Nairobi, Kenya Colony.

G. M. Neal has been made superintendent at Muscoda ore mines of the Tennessee Coal, Iron & Railroad Co., Birmingham. He had been

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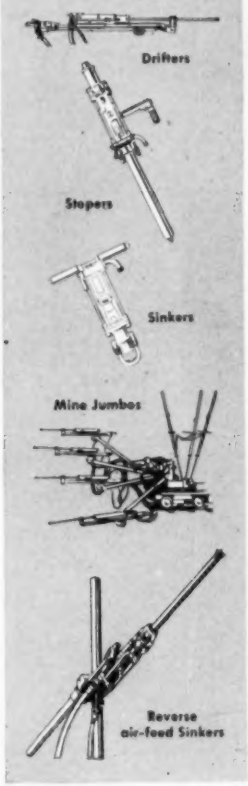

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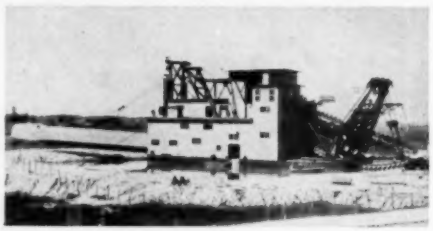
California — YUBA Dredge No. 98, with wood hull, 9 cu. ft. buckets, 30' digging depth, recovering gold for La Grange Gold Dredging Co.



Korea — YUBA Dredge No. 79, steel hull, 10 cu. ft. buckets, 33' digging depth, gold mining.

Yukon — Digging gold near Dawson for Yukon Consolidated Gold Corp. YUBA dredge with wood hull; designed for Arctic conditions. 7 cu. ft. buckets, 25' digging depth.

Montana — Perry & Schroeder Mining Co.'s dredge, built with pontoon-type hull, 6 cu. ft. buckets, and 38' digging depth for mining gold, garnets and sapphires; now being used for backfilling cofferdams at Canyon Ferry Dam site and for excavating aggregate.

Malaya — Kampar Malaya No. 1, with steel hull and 10 cu. ft. buckets, digging tin at depths to 75 ft. below water.

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D. E. Norquist has resigned his position as manager, grinding media sales, with Sheffield Steel Corp., Kansas City, Mo.

Charles A. O'Connell has become assistant mine superintendent at Mufulira Copper Mines Ltd., Mufulira, N. Rhodesia, Africa. He previously held the position of underground manager.

R. H. Rimmer has been elected treasurer of the Chemical Institute of Canada for the year 1950-51. Mr. Rimmer is with Aluminum Laboratories, Ltd., Kingston, Ont., Canada.

Curtis F. Rivers has resigned as general superintendent of coal mines, Tennessee Products & Chemical Corp., Whitwell, Tenn. He is now general manager of the Jackson Equipment Sales, Inc., Middlesboro, Ky. and also a partner.

Robert E. Simpson is now employed as superintendent for the State River Mining Co., Crested Butte, Colo.

Richard E. Seidel is now employed in the purchasing department of the Bethlehem Steel Co., Johnstown, Pa. as a junior buyer.

Neville S. Spence has been appointed director of technical service and development, Dominion Magnesium, Ltd., Toronto, Ont., Canada.

Herman J. Steinbuchel, Jr., is now employed as a mining engineer trainee with the Oliver Iron Mining Co., Hibbing, Minn.

Van B. Stith

Van B. Stith has been appointed vice-president of mines, it was announced by the Anchor Coal Co., Cleveland.

W. Arthur Wall has resigned as plant manager for the Cobalt Chemical & Refining Co., Cobalt, Ont., Canada.

William J. Whatley has accepted the position of chief mechanical engineer with Wyatt C. Hedrick, architect and engineer, Houston. He was formerly mechanical engineer with Freese, Nichols & Turner, consulting engineers, Houston.

C. Campbell White has been appointed to a position in the mill of the Zinc Corp., Broken Hill, New South Wales, Australia.

J. Ward Williams is vice-president of the Masara Consolidated Mines Corp., Manila, P. I. which was formerly Panaminas Inc., Manila.

A. H. Wohlrab is retiring as vice-president and general manager of the Calumet division, Calumet & Hecla Consolidated Copper Co., Calumet, Mich. He will continue as consultant in connection with the zinc operations at Shullsburg, Wis. **O. A. Rockwell,** former assistant manager, has been appointed general manager of the division to succeed Mr. Wohlrab.

Barclay G. Anderson is now working as a mining engineer connected with the U. S. Bureau of Mines, Washington, D. C.

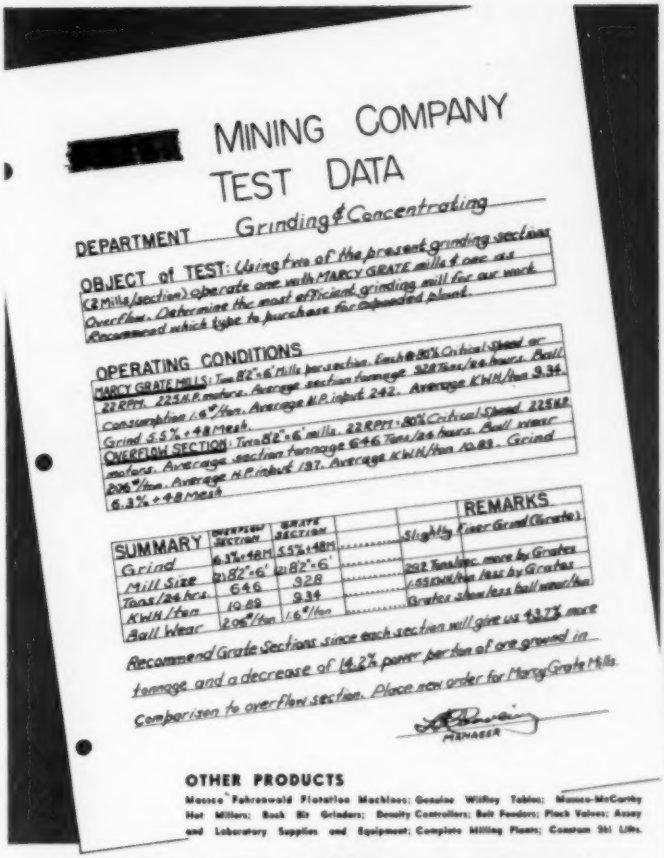
Jesse F. Core has joined the Island Creek Coal Co., Holden, W. Va., as chief engineer. Before joining this company he was employed as chief engineer for the Buckeye Coal Co., Nemacolin, Pa. for two and a half years.

Harold B. Ewoldt is now employed by the Copper Range Co., Boston, Mass. He was formerly general manager for the Calumet Corp., Shullsburg, Wis.

Arno C. Fieldner, chief of the Fuels & Explosives Div., Bureau of Mines, has returned from a visit during the month of July to coal research laboratories in England and France. He participated in the Fourth World Power Conference in London, and the Sixth International Conference of Directors of Mine Safety Research at Verneuil, France.

J. B. H. Fitzwilliam has recently completed his appointment as research fellow at the Missouri School of Mines and Metallurgy. He has taken a position with the Hudson Bay Mining & Smelting Co., Flin Flon, Canada, as a mining engineer.

Kenneth L. Hill recently graduated from the New Mexico School of Mines with the Degree of B. S. in Met. Engineering. He is now employed as a metallurgist by the Calera Mining Co., Forney, Idaho.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OPERATING CONDITIONS

MARCY GRATE MILLS: Two 8'2"×6' Mills per section. Each @ 80% Critical Speed or 22 RPM. 225 H.P. motors. Average section tonnage 328 Tons/24 hours. Ball consumption 1.6 #/Ton. Average H.P. input 242. Average KWH/Ton 9.34. Grind 55% +48 Mesh.

OVERFLOW SECTION: Two 8'2"×6' mills. 22 RPM @ 80% Critical Speed 225 HP motors. Average section tonnage 646 Tons/24 hours. Ball wear 206 #/Ton. Average H.P. input 197. Average KWH/Ton 10.89. Grind 63% +48 Mesh

SUMMARY	OVERFLOW SECTION	GRATE SECTION		REMARKS
Grind	63%+48M	55%+48M	.....	Slightly Finer Ground (Overflow)
Mill Size	2) 8'2"×6'	2) 8'2"×6'	.....	
Tons/24 hrs.	646	328	.....	292 Tons/day more by Grates
KWH/Ton	10.89	9.34	.....	1.55 KWH/Ton less by Grates
Ball Wear	206 #/Ton	1.6 #/Ton	.....	Grates show less ball wear/ton

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Fred J. Hoff is now employed by the International Minerals & Chemical Corp., Lakeland, Fla., as a research engineer.

Cedric L. Iverson is now working as a geologist with the Bureau of Reclamation at Salem, Oregon. He is inspecting the dam-site at Crescent Lake, Oregon, and later will be working on the Benham Falls site south of Bend Ore.

George A. Kiersch was recently transferred to the State Department, International Boundary & Water Commission, Alpine, Texas. He will be supervising geologist under the project engineer of a large geological investigation program along the Rio Grande River throughout the Big Bend country.

John A. McCutchin was transferred from the Calgary site of the Bay Petroleum Corp. to Midland, Texas. He has assumed the duties of division manager.

Joseph H. Merrow, Jr. is now employed as a junior geologist by Patino Mines & Enterprises Consolidated, Inc., Llallagua, Boliva.

Daniel Y. Meschter is presently engaged in manganese and base metals investigations in the Philipsburg district in Montana. He is a geologist with the U. S. Geological Survey.

Pierrepont Arthur Meyer, Jr. is now employed as a field engineer for the St. Joseph Lead Co., Bonne Terre, Mo.

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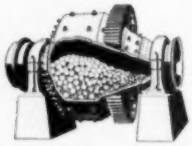

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W. M. Mireylees is leaving for West Africa to take a position with the Amalgamated Banket Areas, Ltd., Tarkwa, Gold Coast Colony. He was formerly residing in Edinburgh, Scotland.

Robert C. Musa, Long Island, N. Y. has been awarded the $2000 fellowship established recently at Columbia University, N. Y., by the Socony-Vacuum Oil Co., Inc. He will work for a doctorate in surface chemistry. Mr. Musa is an alumnus of City College of New York and received his master's degree at Columbia. The fellowship is one of 20 being supported by the Socony-Vacuum laboratories during the 1950-51 academic season at universities and colleges throughout the country. Recipients are selected from students with at least one year of graduate work. No restrictions are placed by the company on the recipients as to future employment.

Sidney Paige, who retired from the Research & Development Board, Department of Defense, Washington, D. C. earlier in the year, will engage in consulting practice in New York City. He may be addressed at Alpine, N. J. or Box 6, Schermerhorn Hall, Columbia University, New York. Mr. Paige is a Visiting Professor of Engineering Geology at Columbia University.

Anthony L. Payne has accepted a position with the Northern Peru Mining & Smelting Co. at the company's Quiruvilca property in Peru. He is employed as resident geologist. After working with the U. S. Geological Survey for three years, Mr. Payne returned to school last year and completed work for the degree of Master of Science at the University of Utah.

Roy E. Peterson is now a graduate student in process metallurgy at the University of California.

P. L. Richards is employed by the H. C. Frick Coke Co., Pittsburgh as a staff engineer in research and preparation.

Thomas J. Williams, president of Sociedad Minera Argentina, Buenos Aires, and **Bruce R. Burnett**, general manager of the same company, have been in the United States in connection with development of the company's sulphur deposit in Argentina. It is in a volcanic cone at a height of 18,000 ft and is said to be the largest deposit of its kind in the world. A contract has been closed with the Chemical Construction Co. for an aerial tramway and mining and treatment equipment, the latter including flotation. Only material exceeding 50 pct sulphur will be mined, but some practically pure sulphur without undesirable impurities was found. The lower grades are mixed with volcanic ash. The product will be marketed in Argentina only. Tungsten and bismuth deposits are also owned by the company. A hundred tons of tungsten concentrates, mixed wolframite and scheelite, can be produced per month, but the property is temporarily closed down for lack of a market.

Abstracts of Technical Papers in Journal of Metals

During the past eight months, several articles have been published in JOURNAL OF METALS, the AIME Metals Branch publication, that bear on the activities of mining site operations. These articles, some which will appear in Metals Transactions and some that appeared in the feature section of JOURNAL OF METALS, are listed below for the convenience of the reader.

Single copies of issues in which individual articles appeared can be obtained from AIME headquarters at the single copy price of the magazine, or photostats of the articles can be obtained from the Engineering Library, at the regular prices for such work.

January: **Morenci Smelter of Phelps Dodge Corp. at Morenci, Ariz.** *L. L. McDaniel.*

March: **Operation, Portsmouth Sinter Plant.** *Guy B. Hunner.*

Graphical Presentation of Metallurgical Equilibria. *C. J. Osborn.* Temperature dependence of free energies of formation of metallurgically important oxides, sulphides, chlorides, carbonates and sulphates is presented graphically, and task of deriving practical metallurgical information is greatly simplified. Diagrams revised according to latest available thermodynamic data.

Conditioning Dwight-Lloyd Gases to Increase Bag Life. *R. E. Shinkoskey.* Outlined development of a program for increasing life of woolen bags used for filtering Dwight-Lloyd gases by treating the bags and gases with hydrated lime. Methods and apparatus are described for determining alkalinity of dusts, acidity and breaking strength of bag cloth. Procedure and results are presented.

An Improved Automatic Smoke Sampler. *J. J. Donoso.*

April: **Use of Adirondack Sinter in Blast Furnaces.** *Elmer H. Riddle.* **Water Sealed Wind Boxes for Dwight and Lloyd Sintering Ma-**

chine. *E. McL. Tittmann and E. A. Hase.* Double roasting of sinter carrying a high percentage of lead concentrates, brought the problem of removing sheets of metallic lead formed in wind boxes. The solution has been found in water sealed wind boxes. The metallic lead is granulated and can be removed without seriously interfering with the operation of the machine.

Air Conditioning of Metallurgical Smoke for Cottrell Precipitation. *A. L. Labbe.* It was found that the presence of free acid in the precipitated dust was essential for efficient operation. This paper deals with the author's experience on the operation of an early Cottrell installation which led to development of a sulphuric acid fuming furnace for auxiliary acid conditioning of combined Wedge roaster and lead sinter smoke.

May: **The Calbeck Process for Refining Zinc Oxide.** *John H. Calbeck, William T. Maidens and Oscar J. Hassel.* The rotary gas fired reheating furnace used by the American Zinc Oxide Co. at Columbus, Ohio, for refining lead-free zinc oxide is described. Outstanding features of this operation are: color of the zinc oxide is greatly improved, sulphur is eliminated, and cadmium recovered without densifying the product to an objectionable degree.

Treatment of Electrolytic Copper Refinery Slimes. *J. H. Schloen and E. M. Elkin.* All known methods of treating and recovering the various components of copper refinery slimes are discussed. The slimes treatment processes presently used by five copper refineries are described and flowsheets are given. A bibliography is appended.

Modern Baghouse Practice for the Recovery of Metallurgical Fumes. *A. L. Labbe and J. J. Donoso.* Hard-won experience in the operation of smelting plants has pointed the way to the most efficient design and economical operation of a baghouse system for recovery of metallurgical fumes. This paper treats of the principles governing bag filtration and the factors most directly affecting bag life. Methods for temperature conditioning of smoke previous to filtering are described and the evolution of bag shaking mechanisms is brought up to date.

June: **A Survey of Thermodynamics of Copper Smelting.** *R. Schuhmann, Jr.* Available thermodynamic data applicable to copper smelting systems are collected and tabulated, and the important gaps are pointed out. A few examples are given of estimations which can be made from the available data. An experimental research program is proposed to supply the thermodynamic data that appear most essential to better quantitative understanding of the chemistry of copper smelting. The proposed program is designed also to shed specific light on the practical problems of slag losses and include magnetite behavior.

Thermodynamic Relationships in Chlorine Metallurgy. *H. H. Kellogg.* Equations representing the standard free energy of formation as a function of temperature, for thirty metallic chlorides, are presented and plotted on a free-energy vs. temperature diagram. The use of these data for calculations on reduction of metallic chlorides, refining of metals with chlorine, and chlorination of metallic oxides and sulphides is illustrated.

July: **Development of Mechanical Puncher at the McGill Smelter.** *Leonard Larson.*

August: **Ferrochromium from Low Grade Chromite Ores and Concentrates.** *F. W. Wessel and R. T. C. Rasmussen.*

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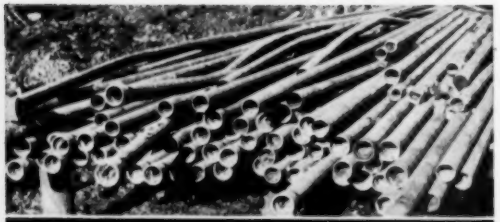
—Obituaries—

Charles B. Carpenter (Member 1938), died suddenly on June 10 at Dallas. He was supervising engineer of the Dallas petroleum field office of the Bureau of Mines, U. S. Dept. of the Interior. He was born at Guthrie, Okla., in 1901 and entered the University of Colorado where he graduated in 1924 with a B.S. in civil engineering. He did engineering work in Oklahoma, Colorado, and Texas prior to joining the Bureau. In 1928 he joined the Bureau of Mines. He had conducted or collaborated in a large number of petroleum engineering studies in various oil fields in Texas and Arkansas. He was appointed supervising engineer of the Dallas office in June 1948. In February and March of this year he served as the United States delegate to the first Mexican Petroleum Convention in Mexico City, where he presided at one of the sessions.

George W. Coffey (Member 1918), died on July 4. He was born in Nevada City, Calif., and received his B.S. in mining engineering from the University of California in 1917. He was a junior engineer on several mining examinations, working with the Miami Copper Co. and Yeatman & Berry. In 1926 he joined the Yukon Gold Co., now Pacific Tin Consolidated Corp., serving as resident manager in the Federated Malay States from 1931 to 1939. He was then named vice-president and in 1945 made chairman and president.

Joseph S. Coupal (Member 1939), former consulting mining engineer, died June 4. Mr. Coupal was state president of the Arizona Small Mine Operators Assn. and president of the organization's Phoenix council. He was born in Lawrence, Mass., in September 1883. He graduated with the degree of B.S. in mining and metallurgy from M.I.T. He was employed by various mining companies on smelter, mine and construction work. Four years were spent in Mexico as mine manager for Providencia Mines Co. and Santa Lucia mines. In 1914 he began his consulting career and was engaged in examination and mine management operations in the western United States, Mexico, Venezuela, and Peru. In 1939, when the Arizona Dept. of Mineral Resources was established, Mr. Coupal was its first director.

Clyde E. Cowan (Member 1936), died in March after an illness of more than a year. He was born at Greensburg, Pa., in 1893. He received the degree of B.A. in 1904 when he graduated from Otterbein College. Following graduation he obtained miscellaneous experience



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¾"	1.050	0.824	350	0.14	400 ft. coils
1"	1.310	1.070	200	0.18	300 ft. coils
1¼"	1.660	1.380	200	0.27	300 ft. coils
1½"	1.900	1.610	200	0.32	250 ft. coils
2"	2.378	2.070	170	0.44	200 ft. coils
3"	3.504	3.070	165	0.91	100 ft. coils
4"	4.504	4.030	150	1.25	25 ft. str.
6"	6.630	6.070	115	2.23	25 ft. str.

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on topography, railroad and reservoir work. In 1909 he was employed by the Jamison Coal Co., Greensburg, Va., as chief engineer. He later joined the South Union Coal Co., Lincoln Gas Coal Co., Brownsville Coal & Coke Co., and Heisely Coal Co., respectively. He joined Heisely as director and operating vice-president.

Octave P. Ebeling (Member 1931), former secretary-treasurer of the South American Development Co., New York, died on January 16. He was born in Liverpool, England, in 1881, but graduated from New York University in 1908. He joined the staff of the South American Development Co. and its holding company, South American Mines Co., and remained with that concern in various capacities. He began as an assistant to various officers of the concern and was closely connected with the management of the company's gold mining enterprise. He later assisted in the selection and engagement of engineers and other employees for the mines in Ecuador.

Charles W. Saxman (Member 1906), retired, has died. He was born on November 22, 1871, at Latrobe, Pa. He spent four years at Harvard University, general science course. For one year he was mining engineer for Montana Coal & Coke Co. He then joined the Idaho Midland Railroad Co., Boise, as chief engineer. Five years were then spent in independent mining in Idaho, British Columbia, Utah and Nevada. In 1905 he was employed by the Greenwich Coal & Coke Co., Latrobe, Pa., as general superintendent.

E. C. Wilson (Member 1915), retired manufacturer, died on June 12. He was born in 1870 at Darke County, Ohio, attended public schools in Los Angeles, and Stanford University for one year. He was with Baker Iron Works, Los Angeles, for approximately seven years. During this time he invented the Wilson patented casing spear and the under-reamer. In 1904 he was manager of Bakersfield Iron Works. He became president of the Wilson Willard Mfg. Co. in 1909 and also invented the Wilson casing elevators. He retired in 1930, but remained in Los Angeles.

Necrology

Date Elected	Name	Date of Death
1945	Harold S. Arnold	Aug. 7, 1950
1893	John F. Berry	Unknown
1927	O. A. Crichett	July 11, 1950
1949	M. G. Driessen	July 16, 1950
1905	Hugh F. Marriott	1949
1906	Warren A. Moller	May 29, 1950
1940	Frank A. Newquist	Mar. 4, 1950
1914	Edwin J. Prindle	Unknown
1947	Herman Shaw	Unknown

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Ray—Kast, Rene F. (C/S—S-J).
Dragoon—Baker, Arthur, III (J) (C/S—S-J).

California

Alpine—Darnell, Balfour F. (R, M).
San Gabriel—Brandt, William O. (R,C/S—S-M).
Los Angeles—Jacobsen, John R. (C/S—S-J).

Colorado

Denver—Hoy, Thomas A. (C/S—S-M).
Golden—Abendschan, John J. (C/S—S-J).

Florida

Bartow—Clawson, Floyd J. (R,C/S—S-J).
Lakeland—Permenter, James C., Jr. (J).

Georgia

Atlanta—Pierson, Richard E. (C/S—S-J).

Idaho

Wallace—Featherstone, Wray W. (C/S—J-M).

Illinois

Chicago—Gracey, Wayne C. (J) (C/S—S-J).
Depue—Morgenstern, Carl G. (M) (R, C/S—J-M).

Indiana

Wabash—Gullings, LeRoy V. (M).

Kansas

Pleasanton—Sarapuu, Erich. (M).

Kentucky

Coalgood—Newsome, Thomas F., Jr. (J).
Hazard—Gibson, Raymond W., Jr. (C/S—S-J).

Michigan

Republic—Gustafson, Allen A. (J).

Minnesota

Coleraine—Knudson, John R. (C/S—S-J).
Hibbing—Herkenhoff, Earl C. (R,C/S-J—J-M).
Nashwauk—McNaughton, David D. (C/S—S-J).

Missouri

Rolla—Drake, Avery A., Jr. (C/S—S-J).
St. Louis—Castelli, August V. (C/S—S-J).

Montana

Butte—Steuer, Neil B. (J).
Butte—Van Matre, Victor Emile (J).

New Jersey
Clifton—Patterson, John A. (R,C/S—S-J).

New Mexico
Hurley—Schroer, Edward A. (M).

New York
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New York—Melitzer, Samuel. (R,C/S—M-A).
West Hempstead—Kyburz, Edward P. (C/S—S-J).

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Rio de Janeiro—Puttonen, Arthur A. (R,C/S—S-M).

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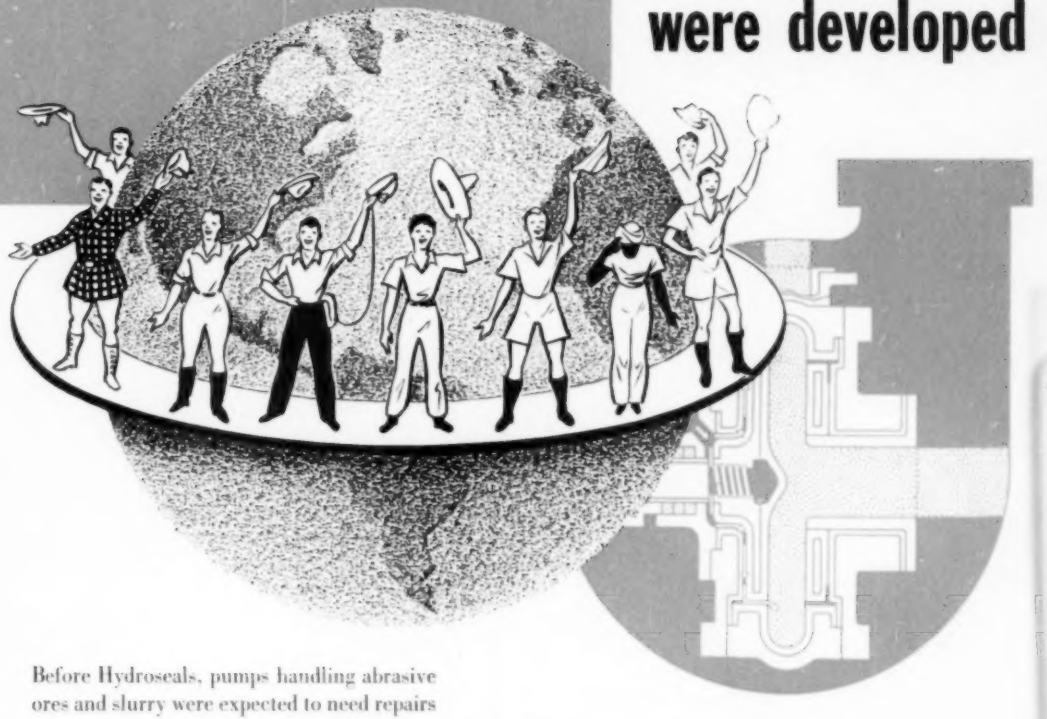

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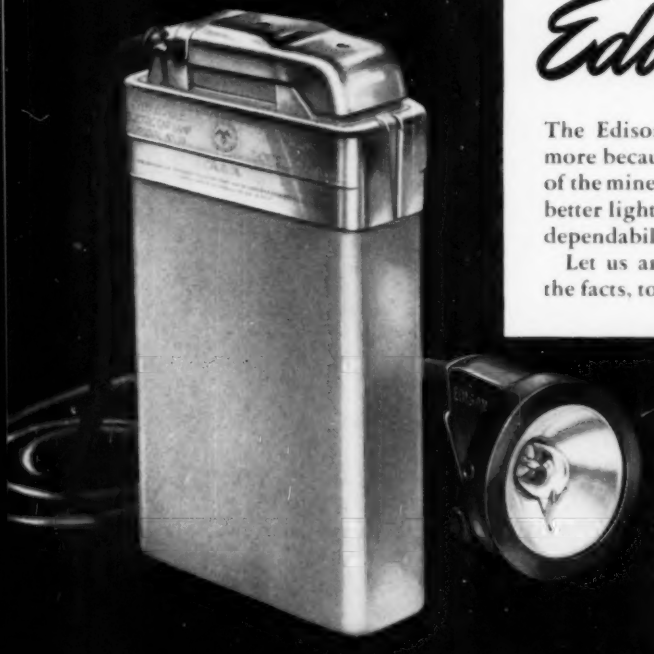

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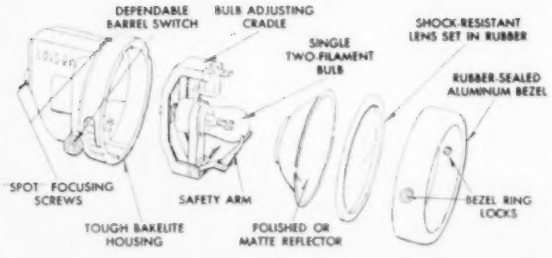

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